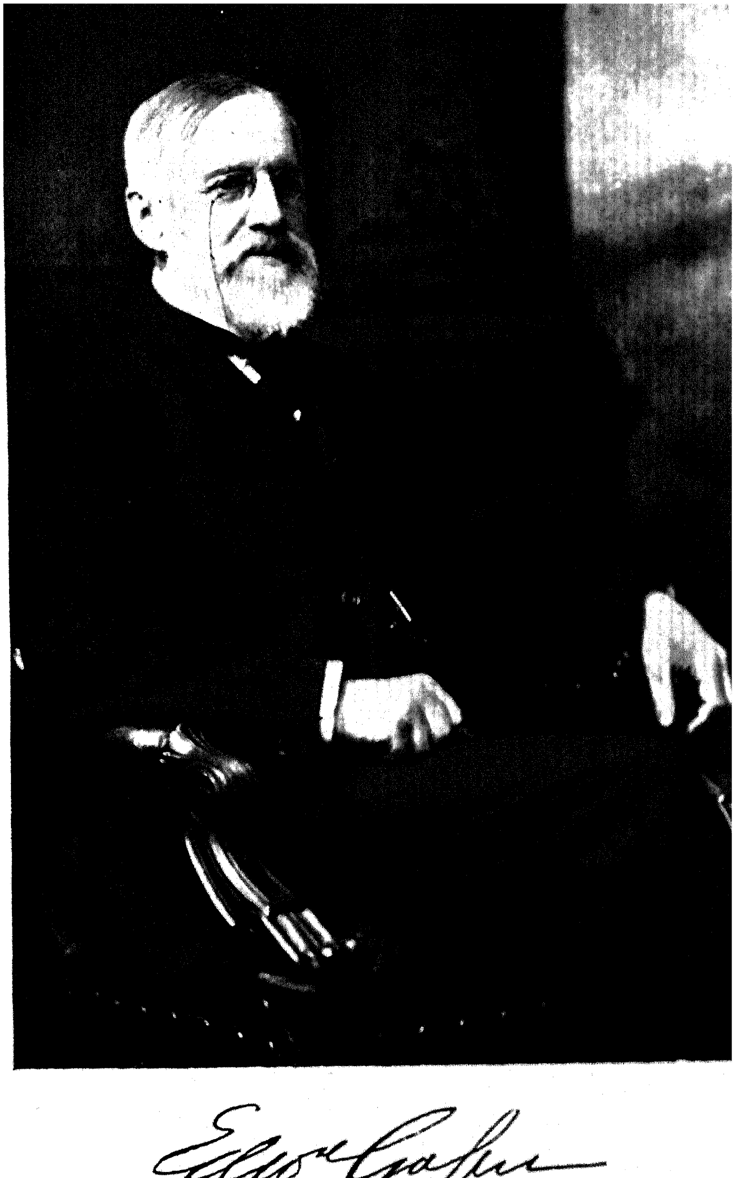

See Biographical Notice, p. 349.

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XXXVII.

CONTAINING THE PAPERS AND DISCUSSIONS OF 1906.

NEW YORK, N. Y.:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1907.

PREFACE.

This volume contains all the published proceedings, papers, and discussions of 1906, with the following exceptions:

1. Certain discussions received early in 1906, but included in Volume XXXVI. of the *Transactions*, since they refer to papers therein contained.

2. Brief obituary notices of Members and Associates reported as deceased during 1905.*

3. Library reports and other announcements of general but temporary interest, furnished to members in the six numbers of the *Bulletin* for the year 1906.

4. Certain valuable papers presented in joint session with the Iron and Steel Institute at the London Meeting, July, 1906, and, under a mutual agreement between the Councils of the two Institutes, available for publication by the American Institute of Mining Engineers. These papers have been published in the *Bulletin*.†

5. Addresses of welcome by R. A. Hadfield, President, and Sir James Kitson, past-President, of the Iron and Steel Institute, at the London Meeting,‡ and reply by President R. W. Hunt.§

6. Accounts of the various excursions and entertainments in England, Scotland, Wales and Germany.**

7. List of Members and Associates revised to Jan. 1, 1907. Published in separate pamphlet form and distributed with *Bi-Monthly Bulletin*, No. 13, January, 1907.

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* *Bi-Monthly Bulletin*, No. 9, May, 1906, pp. 359 to 378.

† The Influence of Silicon and Graphite on the Open-Hearth Process, by Alex. S. Thomas, *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 931 to 938 ; The Constitution of Iron-Carbon Alloys, by Albert Sauveur, *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 939 to 966 ; The Kjellin Electric Steel Furnace, by E. C. Ibbotson, *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 967 to 970.

‡ *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 816 to 824.

§ *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 824 to 825.

** *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 860 to 907.

The publications thus mentioned above amount to about 400 pages of valuable material which it was impracticable to include in this volume of the *Transactions* on account of lack of space.

On the other hand, this volume includes certain discussions referring to papers contained in it, which were received after Jan. 1, 1907, yet sufficiently early to be here printed instead of being held over for Volume XXXVIII.

JOSEPH STRUTHERS,
Assistant Secretary and Editor.

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DISCUSSIONS.

OFFICERS.

For the year ending February, 1907.

COUNCIL.*

PRESIDENT OF THE COUNCIL.

ROBERT W. HUNT......CHICAGO, ILL.

(Term expires February, 1907.)

VICE-PRESIDENTS OF THE COUNCIL.

WILLIAM P. BLAKE......TUCSON, ARIZ.
THOMAS F. COLE......DULUTH, MINN.
IRVING A. STEARNS......WILKES-BARRE, PA.

(Term expires February, 1907.)

HENRY M. HOWE......NEW YORK, N. Y.
J. B. GRANT......DENVER, COLO.
JAMES D. HAGUE......NEW YORK, N. Y.

(Term expires February, 1908.)

COUNCILORS.

F. L. GRAMMER......BALTIMORE, MD.
JOSEPH HARTSHORNE......POTTSTOWN, PA.
CHARLES H. SNOW......NEW YORK, N. Y.

(Term expires February, 1907.)

A. A. BLOW......WASHINGTON, D. C.
FRANK LYMAN......NEW YORK, N. Y.
T. A. RICKARD......SAN FRANCISCO, CAL.

(Term expires February, 1908.)

THEODORE DWIGHT......NEW YORK, N. Y.
WALTER WOOD......PHILADELPHIA, PA.
WM. FLEET ROBERTSON......VICTORIA, B. C., CAN.

(Term expires February, 1909.)

SECRETARY OF THE COUNCIL.

R. W. RAYMOND......NEW YORK, N. Y.

(Term expires February, 1907.)

ASSISTANT SECRETARY AND EDITOR.

JOSEPH STRUTHERS......NEW YORK, N. Y.

CORPORATION.

JAMES GAYLEY, President ; JAMES DOUGLAS, Vice-President ;
R. W. RAYMOND, Secretary ; FRANK LYMAN, Treasurer ;
H. W. B. HOWARD, Assistant Secretary and Assistant Treasurer.

DIRECTORS.

JAMES GAYLEY, FRANK KLEPETKO, FRANK LYMAN.

(Term expires February, 1907.)

JAMES DOUGLAS, JAMES F. KEMP, ALBERT R. LEDOUX.

(Term expires February, 1908.)

THEODORE DWIGHT, CHARLES H. SNOW, R. W. RAYMOND.

(Term expires February, 1909.)

Consulting Attorneys, Blair & Rudd, New York, N. Y.

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

POSTSCRIPT.

The list of officers on the preceding page is that of the year 1906, which is the period covered by the contents of this volume. But the result of the election held at the Annual Meeting of February, 1907, although strictly belonging to the next volume, is here published for the convenience of members.

The following officers were elected at the Annual Meeting, Feb. 19, 1907:

COUNCIL.

PRESIDENT: John Hays Hammond, New York, N. Y.

VICE-PRESIDENTS: *For two years* (term expires February, 1909), Samuel B. Christy, Berkeley, Cal.; John A. Church, New York, N. Y.; Persifor Frazer, Philadelphia, Pa.

SECRETARY: R. W. Raymond, New York, N. Y.

ASSISTANT SECRETARY (by appointment): Joseph Struthers, New York, N. Y.

COUNCILORS: *For three years* (term expires February, 1910), B. F. Fackenthal, Jr., Easton, Pa.; H. O. Hofman, Boston, Mass.; Walter R. Ingalls, New York, N. Y.

CORPORATION.

PRESIDENT: James Gayley, New York, N. Y.

VICE-PRESIDENT: James Douglas, New York, N. Y.

SECRETARY: R. W. Raymond, New York, N. Y.

TREASURER: Frank Lyman, New York, N. Y.

ASSISTANT SECRETARY AND ASSISTANT TREASURER (by appointment): Joseph Struthers, New York, N. Y.

DIRECTORS: *For three years* (term expires February, 1910), James Gayley, New York, N. Y.; Charles Kirchhoff, New York, N. Y.; Frank Lyman, New York, N. Y.

PAST OFFICERS.

Presidents.

*David Thomas........1871
R. W. Raymond........1872
R. W. Raymond........1873
R. W. Raymond........1874
*A. L. Holley........1875
*Abram S. Hewitt........1876
*T. Sterry Hunt........1877
*Eckley B. Coxe........1878
*Eckley B. Coxe........1879
*William P. Shinn........1880
William Metcalf........1881
*Richard P. Rothwell........1882
Robert W. Hunt........1883
James C. Bayles........1884
James C. Bayles........1885
Robert H. Richards........1886
*Thomas Egleston........1887
William B. Potter........1888
Richard Pearce........1889
*Abram S. Hewitt........1890
John Birkinbine........1891
John Birkinbine........1892
H. M. Howe........1893
John Fritz........1894
*J. D. Weeks........1895
E. G. Spilsbury........1896
*Thomas M. Drown........1897
C. Kirchhoff........1898
James Douglas........1899
James Douglas........1900
E. E. Olcott........1901
E. E. Olcott........1902
Albert R. Ledoux........1903
Albert R. Ledoux........1904
R. W. Hunt (Council)........1905
James Gayley (Corporation)........1905

Secretaries.

*Martin Coryell........1871–1872
*Thomas M. Drown........1873–1884
R. W. Raymond........1884 ——

Treasurers.

J. Pryor Williamson........1871–1872
*Theodore D. Rand........1872–1903
Frank Lyman........1903 ——

* Deceased.

HONORARY MEMBERS.

Prof. Richard Åkerman	Stockholm, Sweden.
Andrew Carnegie	New York, N. Y.
Dr. James Douglas	New York, N. Y.
R. A. Hadfield	Sheffield, England.
Prof. Hans Hoefer	Leoben, Austria.
Prof. Haton de la Goupillière	Pau, Basses, Pyrénées, France.
Prof. Henri Louis Le Chatelier	Paris, France.
M. Floris Osmond	Paris, France.
John E. Stead	Middlesbrough, England.
Prof. Dimitry Constantin Tschernoff	St. Petersburg, Russia.
Prof. Dr. Hermann Wedding	Berlin, Germany.

Honorary Members (*Deceased*).

Bell, Sir Lowthian	1904
Castillo, A. Del	1895
Contreras, Manuel Maria	1902
Daubrée, A	1896
Drown, Thomas M	1904
Gaetzschmann, Moritz	1895
Gruner, L	1883
Hunt, T. Sterry	1892
Kerl, Bruno	1905
Le Conte, Joseph	1901
Lesley, J. P.	1896
Patera, Adolph	1890
Percy, John	1889
Posepny, Franz	1895
Richter, Theodor	1898
Roberts-Austen, W. C.	1902
Serlo, Albert	1898
Siemens, C. Williams	1883
Thomas, David	1882
Tunner, Peter R. von	1897

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO JULY, 1906.

Number.	Place.	Date.	Transactions. Vol.	Page.
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II.	Bethlehem, Pa	August, 1871	i.	10
III.	Troy, N. Y	November, 1871	i.	13
IV.	Philadelphia, Pa	February, 1872	i.	17
V.	New York, N. Y.*	May, 1872	i.	20
VI.	Pittsburg, Pa	October, 1872	i.	25
VII.	Boston, Mass	February, 1873	i.	28
VIII.	Philadelphia, Pa.*	May, 1873	ii.	3
IX.	Easton, Pa	October, 1873	ii.	7
X.	New York, N. Y	February, 1874	ii.	11
XI.	St. Louis, Mo.*	May, 1874	iii.	3
XII.	Hazleton, Pa	October, 1874	iii.	8
XIII.	New Haven, Conn	February, 1875	iii.	15
XIV.	Dover, N. J.*	May, 1875	iv.	3
XV.	Cleveland, O	October, 1875	iv.	9
XVI.	Washington, D. C	February, 1876	iv.	18
XVII.	Philadelphia, Pa.†	June, 1876,	v.	3
XVIII.	Philadelphia, Pa	October, 1876	v.	19
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XX.	Wilkes-Barre, Pa.*	May, 1877	vi.	3
XXI.	Amenia, N. Y	October, 1877	vi.	10
XXII.	Philadelphia, Pa	February, 1878	vi.	18
XXIII.	Chattanooga, Tenn.*	May, 1878	vii.	3
XXIV.	Lake George, N. Y	October, 1878	vii.	103
XXV.	Baltimore, Md.*	February, 1879	vii.	217
XXVI.	Pittsburg, Pa	May, 1879	viii.	3
XXVII.	Montreal, Canada	September, 1879	viii.	121
XXVIII.	New York, N. Y.*	February, 1880	viii.	275
XXIX.	Lake Superior, Mich	August, 1880	ix.	1
XXX.	Philadelphia, Pa.*	February, 1881	ix.	275
XXXI.	Staunton, Va	May, 1881	x.	1
XXXII.	Harrisburg, Pa	October, 1881	x.	119
XXXIII.	Washington, D. C.*	February, 1882	x.	225
XXXIV.	Denver, Col	August, 1882	xi.	1
XXXV.	Boston, Mass.*	February, 1883	xi.	217
XXXVI.	Roanoke, Va	June, 1883	xii.	3
XXXVII.	Troy, N. Y	October, 1883	xii.	175
XXXVIII.	Cincinnati, O.*	February, 1884	xii.	447
XXXIX.	Chicago, Ill	May, 1884	xiii.	1
XL.	Philadelphia, Pa	September, 1884	xiii.	285
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XLII.	Chattanooga, Tenn	May, 1885	xiv.	1

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

Number.	Place.	Date.	Transactions. Vol.	Page.
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XLIV.	Pittsburg, Pa.*	February, 1886	xiv.	587
XLV.	Bethlehem, Pa.	May, 1886	xv.	lxiii.
XLVI.	St. Louis, Mo.	October, 1886	xv.	lxx.
XLVII.	Scranton, Pa.*	February, 1887	xv.	lxxvii.
XLVIII.	Utah and Montana	July, 1887	xvi.	xvii.
XLIX.	Duluth, Minn.	July, 1887	xvi.	xxiv.
L.	Boston, Mass.*	February, 1888	xvi.	xxviii.
LI.	Birmingham, Ala.	May, 1888	xvii.	xix.
LII.	Buffalo, N. Y.	October, 1888	xvii.	xxiv.
LIII.	New York, N. Y.*	February, 1889	xvii.	xxxi.
LIV.	Colorado	June, 1889	xviii.	xvii.
LV.	Ottawa, Canada	October, 1889	xviii.	xxiv.
LVI.	Washington, D. C.*	February, 1890	xviii.	xxx.
LVII.	New York, N. Y.	September, 1890	xix.	vii.
LVIII.	New York, N. Y.*	February, 1891	xix.	xxv.
LIX.	Cleveland, O.	June, 1891	xx.	xvi.
LX.	Glen Summit, Pa.	October, 1891	xx.	lxi.
LXI.	Baltimore, Md.*	February, 1892	xxi.	xix.
LXII.	Plattsburgh, N. Y.	June, 1892	xxi.	xxxiii.
LXIII.	Reading, Pa.	October, 1892	xxi.	xliv.
LXIV.	Montreal, Canada*	February, 1893	xxi.	lii.
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LXVI.	Virginia Beach, Va.*	February, 1894	xxiv.	xvii.
LXVII.	Bridgeport, Conn.	October, 1894	xxiv.	xxxv.
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LXXIII.	Lake Superior	July, 1897	xxvii.	xxx.
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LXXVII.	California	September, 1899	xxix.	xlix.
LXXVIII.	Washington, D. C.*	February, 1900	xxx.	xix.
LXXIX.	Canada	August, 1900	xxx.	xlv.
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LXXXII.	Philadelphia, Pa.‡	May, 1902	xxxiii., xxi. and xxxv.	
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LXXXIV.	Albany, N. Y.*	February, 1903	xxxiv.	xxiii.
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XC.	Bethlehem, Pa.	February, 1906	xxxvii.	xli.
XCI.	London, England	July, 1906	xxxvii.	xlviii.

* Annual meeting for the election of officers.

† Begun in February at New York City, for the election of officers, and adjourned to Florida.

‡ " " " " " " " " " " " " " to Philadelphia.

PUBLICATIONS.

THE publications of the Institute comprise:

TRANSACTIONS.

The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there are no important changes.) These volumes are for sale as follows, in paper covers:

Vols. I. to IV., inclusive, each,	$3.00
Vols. V. to VIII., inclusive, each,	4.00
Vols. IX. and X. (a small supply only on hand), each,	10.00
Vols. XI. to XXIX., inclusive, each,	5.00
Vols. XXX. and XXXI., each,	6.00
Vol. XXXII.,	5.00
Vols. XXXIII. to XXXVII., inclusive, each,	6.00
Half-morocco binding, $1 extra per volume.	
Complete set of *Transactions*, Vols. I. to XXXVII., inclusive, half-morocco binding (freight-prepaid),	226.00

INDEXES AND SPECIAL EDITIONS.

Index, Vols. XVI. to XX., inclusive, paper,	$1.00
Index, Vols. XXI. to XXV., inclusive, cloth,	1.25
Index, Vols. XXVI. to XXX., inclusive, cloth, $1.50, ½ mor.,	2.50
Index, Vols. I. to XXXV., inclusive, cloth, $5.00, ½ mor.,	6.00

"*The Genesis of Ore-Deposits*," comprising the famous treatise of the late Professor Franz Posepny, with the successive discussions thereof by Le Conte, Blake, Winchell, Church, Emmons, Becker, Cazin, Rickard and Raymond (all of which were published in Volumes XXIII. and XXIV. of the *Transactions* of the Institute, and subse-

quently in the special "Posepny Volume," issued by the Institute); also, later papers by Van Hise, Emmons, Weed, Lindgren, Vogt, Kemp, Blake, Rickard and others, and the discussions of these papers by De Launay, Beck, and many others (some of these were included in Volume XXX. and the remainder appeared in Volume XXXI.); also a complete bibliography of the Institute papers and discussions on this subject from 1871 to the present time.

The original Posepny volume comprised 265 pages, and was sold for $2.50, at which price the edition was long since exhausted. The present volume is an octavo of 825 pages, bound in "book-linen," $6.00

Half-morocco bound copies, ... 7.00

"*The Evolution of Mine-Surveying Instruments.*" This is a volume of about 400 pages, issued in the same style as the foregoing, and containing the original paper of Mr. Dunbar D. Scott on that subject (*Transactions*, XXVIII.), first published in 1898, together with later papers, continuing the same subject, and discussions thereof, by Hoskold, Lyman, Davis and many others, . 3.50

Half-morocco binding, ... 4.50

Glossary of Mining and Metallurgical Terms (1881), cloth, . 0.50

Spanish-American Mining and Metallurgical Glossary, bound in leather, pocket-size, 96 pages, ... 0.75

Special Mining and Railway Map of Mexico, size 14 by 20, prepared by order of Dept. of Fomento, 1901, ... 0.35

List of Members, Rules, etc., paper, ... 0.50

Bi-Monthly Bulletin, paper (each), ... 2.00

PAMPHLETS.

1. The Minutes of the Proceedings of each Meeting.
2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the edition distributed, without charge, to members and associates not in arrears, a small supply is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880. These papers are for sale at the office of the Secretary, or are sent to purchasers, charges paid, on receipt of the price, as follows:

No. of Pages.	Single Copies.	10 Copies.	20 Copies.
24 pp. or less........................	$0.25	$2.00	$3.50
25 to 48................................	0.30	2.50	4.50
49 to 80................................	0.40	3.25	5.25
81 to 96................................	0.45	3.50	6.00
97 to 128..............................	0.50	3.75	6.25
129 to 150.....	0.55	4.00	6.50

Papers with folders and inserted plates subject to special price.

Authors' Edition of Pamphlets.

Extra copies, if ordered before the printing of the pamphlet edition for the *Bi-Monthly Bulletin*, will be furnished to the members of the Institute at special rates, which will be stated on application to the Assistant Secretary, Joseph Struthers, 29 West 39th Street, New York, N. Y.

CONSTITUTION.

[ADOPTED FEB. 21, 1905.]

ARTICLE I.

NAME AND OBJECT.

SEC. 1. This Institute is incorporated under the Membership Corporation Law of the State of New York; its corporate name is AMERICAN INSTITUTE OF MINING ENGINEERS; and its objects are such as are stated in its Certificate of Incorporation.

ARTICLE II.

MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: (1) Members; (2) Honorary Members; (3) Associates; and (4) Honorary Associates. Only Members and Associates residing within the United States of America, Republic of Mexico and Dominion of Canada shall be entitled to vote at the meetings of the Institute.

SEC. 2. All Members, Honorary Members, Associates and Honorary Associates of the American Institute of Mining Engineers as the same existed on the day of the incorporation of this Institute, are Members, Honorary Members, Associates and Honorary Associates, respectively, of this Corporation.

SEC. 3. The following classes of persons shall be eligible for membership in the Institute, namely: as Members and Honorary Members, all professional mining engineers, geologists, metallurgists or chemists, and all persons practically engaged in mining, metallurgy or metallurgical engineering; as Associates and Honorary Associates, all persons desirous of being connected with the Institute who, in the opinion of the Council, are suitable.

SEC. 4. Every candidate for election as a Member or Associate of the Institute must be proposed for election by at least three Members or Associates; must be approved by the Committee on Membership, as prescribed in the By-Laws; and must be elected by the Council. Not less than three-fourths of the votes cast shall be necessary to an election. Every person so elected shall become a Member or Associate, as the case may be, upon payment of his first dues as hereinafter prescribed. Each candidate for Honorary Member or Honorary Associate, must be recommended by at least ten Members or Associates; must be approved by the Council; and must be elected by ballot at a meeting of the Board of Directors by the unanimous vote of all the Directors present; provided, however, that the number of Honorary Members and Honorary Associates shall not at any time exceed twenty.

SEC. 5. If any person elected a Member or Associate does not, within sixty days after notice of his election, accept the same and pay his initiation fee and dues for the current year, his election may be cancelled at the discretion of the Council.

SEC. 6. The Council may at any time change the classification of a person elected as an Associate so as to make him a Member, or vice versa. All Members and Associates shall be equally entitled to the privileges of membership, provided that Honorary Members, Honorary Associates, and Members and Associates whose Post-Office addresses shall be outside of the United States, Mexico and Canada, shall not be entitled to vote.

ARTICLE III.

DUES.

SEC. 1. The dues of Members and Associates shall be Ten Dollars per annum, payable in advance on the first day of each Calendar year. Each newly elected Member or Associate shall pay, when notified of election, an initiation fee of Ten Dollars in addition to the dues for the current year. Honorary Members and Honorary Associates shall not be liable to initiation fee or dues. Any Member or Associate in arrears for one year may, at the discretion of the Council, be deprived of the receipt of publications or stricken from the list of Members, provided that he may be restored to membership by the Council on payment of all arrears or may be again proposed and elected after an interval of three years.

SEC. 2. Any Member or Associate not in arrears may become, by the payment of One Hundred and Fifty Dollars at one time, a Life Member or Associate; and shall not be liable thereafter to annual dues.

ARTICLE IV.

BUSINESS MEETINGS OF THE INSTITUTE.

SEC. 1. The annual meeting of the Institute for the election of Directors and transaction of other business shall take place on the third Tuesday in February in each year. A report of the financial condition of the Institute and an abstract of the accounts shall be furnished by the Directors, and presented at each annual meeting.

SEC. 2. Special business meetings of the Institute may be held at such times and places as the Board of Directors may appoint, upon notice to all Members and Associates entitled to vote, directed to each at his last known Post-Office address, and mailed in the City of New York not less than twenty days before the date fixed for such meeting.

SEC. 3. At all business meetings of the Institute the presence of nine Members and Associates shall constitute a quorum.

SEC. 4. At all business meetings of the Institute Members and Associates may vote either in person or by proxy, but no Member or Associate in arrears since the last annual meeting shall be entitled to vote.

ARTICLE V.

OTHER MEETINGS OF THE INSTITUTE.

SEC. 1. All meetings of the Institute other than business meetings shall be held at such times and places as the Council may appoint. Notice of all such meetings shall be given to all Members and Associates by mail.

ARTICLE VI.

Directors and Officers.

Sec. 1. The business and financial affairs of the Institute shall be managed by a Board of Directors, who shall be elected at the annual meeting in the manner prescribed in the Certificate of Incorporation.

Sec. 2. The officers of the corporation shall be a President, Vice-President, Secretary and Treasurer, who shall be elected by the Directors from among their number. All such officers shall be elected at the first meeting of the Board of Directors after each annual meeting of the corporation, and shall hold office for one year or until their successors are elected and qualify.

The duties of all officers shall be such as usually pertain to their offices, respectively, together with such other duties as may from time to time be prescribed for them by the By-Laws. The Treasurer shall give a bond for the faithful performance of his duties in a sum to be fixed by the Board of Directors, but at the expense of the Institute.

Sec. 3. In the event of a vacancy occurring in the Board of Directors by death, resignation or otherwise, the remaining members of the Board may, by a majority vote, elect a successor to fill the vacancy, who shall continue in office until the next annual meeting or until his successor shall have been chosen.

Sec. 4. The Board of Directors may, in its discretion, declare the place of any Director vacant, on his failure for any reason, to attend three successive meetings of the Board. Any Director who shall under this section or in any other manner cease to be a member of the Board shall, at the same time, be held to have vacated any other office to which he shall previously have been elected; and the Board shall elect a new incumbent to the said vacant office.

Sec. 5. The Board of Directors may from time to time appoint from their own number standing and special committees, and may delegate to such committees such duties as they may see fit.

ARTICLE VII.

Meetings of the Board of Directors.

Sec. 1. A regular meeting of the Board of Directors for the election of officers and the transaction of other business shall be held on the third Tuesday in February in each year, after the adjournment of the annual meeting of the Institute.

Sec. 2. Special meetings of the Board of Directors, at which any business may be transacted, may be called to meet at any time at the office of the Institute in the City of New York, by notice in writing mailed at least five days before the meeting, by the Secretary to each member of the Board at his last known Post-Office address, signed either by the President or the Vice-President or by three members of the Board.

Sec. 3. At all meetings of the Board of Directors the presence of five members shall constitute a quorum.

ARTICLE VIII.

The Council.

Sec. 1. The professional, technical, scientific and social interests of the Institute shall be committed to the supervision of a Council composed of a President

of the Council, six Vice-Presidents of the Council, a Secretary of the Council and nine Councilors, who shall be elected from among the Members and Associates of the Institute in the manner hereinafter prescribed. Members of the Council may or may not be members of the Board of Directors.

SEC. 2. The President of the Council shall be elected for one year, and no person shall be eligible for immediate re-election to this office who shall have held the same for two consecutive years.

After the first year Vice-Presidents of the Council shall be elected to serve for two years, and Councilors shall be elected to serve for three years. No Vice-President of the Council or Councilor shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. The Secretary of the Council shall be elected annually.

SEC. 3. At the first annual meeting, to be held in the year 1905, there shall be elected a President of the Council to serve for one year, a Secretary of the Council to serve for one year, three Vice-Presidents of the Council to serve for one year, three Vice-Presidents of the Council to serve for two years, three Councilors to serve for one year, three Councilors to serve for two years, and three Councilors to serve for three years. At each subsequent annual meeting there shall be elected a President of the Council to serve for one year; a Secretary of the Council to serve for one year; three Vice-Presidents of the Council to serve for two years; and three Councilors to serve for three years. The term of office of all Members of the Council shall continue until the adjournment of the meeting at which their successors are elected.

SEC. 4. Vacancies in the Council may occur by death or resignation; or the Council may, by the vote of a majority of all its members, declare the place of any officer or member of the Council vacant, on his failure for one year, from inability or otherwise, to attend the regular meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *provided* that the said appointment shall not render such person ineligible for election to the Council at the next meeting.

SEC. 5. The presence of five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or any business coming within the authority of the Council may be transacted at a regularly-called meeting thereof, at which less than a quorum may be present, subject to the approval of a majority of the Council subsequently given in writing to the Secretary and recorded by him with the minutes.

SEC. 6. The election of the Council shall take place at the regular annual meeting of the Institute. Nominations for members of the Council may be sent in writing to the Secretary accompanied with the names of the proposers at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before said meeting, mail to every Member or Associate entitled to vote a list of all nominations for each office so received, together with the names of the persons ineligible for election to each office; and if the Council or a Committee thereof, appointed for the purpose, shall have recommended any nomination, such recommendation may also be sent to the Members and Associates with the list of all nominations made.

ARTICLE IX.

MEETINGS OF THE COUNCIL.

SEC. 1. Meetings of the Council shall be held at such times and places as the President of the Council or one of the Vice-Presidents of the Council may appoint.

SEC. 2. A meeting of the Council may be held on the day of the annual meeting of the Institute without previous notice. Written notice of all other meetings of the Council, specifying the time and place of such meeting, signed by the Secretary, shall be mailed to every member of the Council at his last known Post-Office address at least ten days before the date of the meeting.

ARTICLE X.

PAPERS AND PUBLICATIONS.

SEC. 1. The Council shall have power to decide as to the acceptance and publication of any professional papers presented to the Institute, subject to such conditions as the Board of Directors may prescribe.

SEC. 2. The copyright of all professional papers communicated to and accepted by the Institute shall be vested in it, unless otherwise expressly agreed between the Council and the author. The Institute shall not assume responsibility for any statements of fact or opinion advanced in the papers or discussions at its meetings. Neither the Council nor the Institute shall officially approve or disapprove any technical or scientific opinion or any proposed enterprise, outside of the management of the meetings, discussions and publications of the Institute, and the conduct of its business affairs by the Board of Directors.

SEC. 3. Special Committees may from time to time be appointed by the Council to make investigations and prepare reports for presentation to the Institute, but no action shall be taken binding the Institute for or against the conclusions embodied in any such reports.

ARTICLE XI.

SUSPENSIONS AND EXPULSIONS.

SEC. 1. Any member of the Institute who shall be convicted of a crime involving, in the opinion of the Board of Directors, moral turpitude, shall, upon the passage by the Board of Directors of a resolution declaring the crime for which he has been convicted to be of such character, be thereupon dropped from membership in this Institute.

SEC. 2. Any member of the Institute may be suspended or expelled for misconduct by the Board of Directors, after charges setting forth such misconduct shall have been prepared by the Council and filed in writing with the Board. Upon the receipt of such charges in writing, the Board may, in its discretion, suspend such member pending a hearing and determination thereupon. As soon as may be after the receipt of such charges, the Board shall fix a date for a hearing thereupon and shall give to the accused member notice thereof in writing, mailed to him at his last known Post-Office address not less than thirty days before said date, accompanied by a full copy of the charges and a copy of the second, third and fourth sections of this article.

SEC. 3. Upon the day fixed for the hearing, the accused member may appear before the Board, either in person or by an accredited representative ; hear any

witnesses who may be called in support of the charges and at his option cross-examine the same; and hear read any documentary evidence offered in support of the charges. The accused may, in his discretion, produce and examine witnesses in his defence, and submit documentary evidence, including a statement from himself in writing. After the conclusion of the hearing, the Board of Directors shall consider and vote to approve or disapprove the charges. If the Board shall, by a vote of two-thirds of its members, declare the charges sustained, it may suspend the member for a stated period or expel him.

SEC. 4. If the accused member shall not appear at the hearing, and shall within three months thereafter file with the Board an affidavit stating that he had not received notice of the charges against him in time to enable him to present his defence, the Board shall fix a date for a re-hearing within three months from the receipt of such affidavit and shall immediately notify the accused member by mail of such date. Upon the re-hearing, the accused shall have the same privilege of presenting his defence as he would have had upon the original hearing; and after the defence is presented, the Board shall take a new vote upon the charges, the result of which shall be conclusive.

SEC. 5. All interests in the property of the Institute of persons resigning, or otherwise ceasing to be Members or Associates, shall vest in the Institute.

ARTICLE XII.

AMENDMENTS.

SEC. 1. This Constitution or any Article or Section thereof may be amended at any annual meeting by a two-thirds vote of all the members present in person or by proxy, *provided* that notice of the proposed amendment shall have been given in writing at a previous meeting, and *provided also* that the amendment or amendments so adopted shall have been printed and mailed to all Members and Associates not later than thirty days before the annual meeting. Any amendment or amendments approved by a majority of the votes cast shall be deemed to have been adopted, and shall become a part of this Constitution. The Secretary shall forthwith print and distribute to Members and Associates an announcement of the result of said vote, and if any amendment or amendments shall have been adopted, a copy of the section or sections so amended.

BY-LAWS.

[Adopted Feb. 21, 1905. Amended Feb. 20, 1906, and Nov. 16, 1906.]

I. Presiding Officers.

At all Business meetings of the Institute the President, or, in his absence, the Vice-President, or, in the absence of both of them, any other member of the Board of Directors to be chosen by the meeting, shall preside.

At all other meetings of the Institute the President of the Council or, in his absence, one of the Vice-Presidents, if present, shall preside.

II. Order of Business.

At each Business meeting of the Institute the order of business shall be as follows:

1. Reading of minutes of preceding meeting.
2. Report of the President.
3. Report of the Treasurer.
4. Report of the Secretary.
5. Election of Directors.
6. Election of Members of the Council.
7. Reports of Standing Committees.
8. Reports of Special Committees.
9. Special Orders.
10. Miscellaneous business.

This order of business may be changed by a vote of a majority of the Members and Associates present in person or by proxy.

The usual parliamentary rules shall govern all meetings of the Institute except in cases otherwise provided by the Constitution or the By-Laws.

At all sessions of the Institute other than business meetings, the order of proceedings and the time of adjournment shall rest in the discretion of the presiding officer.

III. Secretary.

The Secretary shall keep a record of the proceedings of all meetings of the Institute. He shall be custodian of the Corporate Seal, of the Minute Books, and of all Legal Documents belonging to the Institute. He shall conduct, on behalf of the Institute, all correspondence relating to business matters, except such as pertains directly to the office of the Treasurer.

He shall notify all officers and Directors and Members of the Council, and all Members of Committees of their election and appointment; shall issue notices of all meetings of the Board, and of the annual and other meetings of the Institute; and shall, in calling special meetings of the Directors, specify the object of such meeting.

IV. Secretary of the Council.

The Secretary of the Council shall act as the Clerk of that body at all of its meetings and at all meetings of the Institute called for the discussion of professional, technical or scientific matters, or for any other purpose than the transaction of business.

He shall be custodian of all technical or scientific papers submitted to the In-

stitute for its consideration, shall have charge of the editing and printing of all material published by the Institute, and of the distribution thereof. On the first day of May following the year in which each volume of *Transactions* is printed, he shall turn over to the Library Committee all copies of the same not theretofore distributed by him. He shall have charge of all the correspondence of the Institute relating to other than business affairs.

The Secretary of the Council shall receive a salary to be fixed by the Board of Directors. He may appoint an Assistant with the title of Editor, who shall likewise receive a salary to be fixed by the Board of Directors.

The Secretary of the Council may or may not be the same person as the Secretary of the Institute.

V. Assistant Secretary.

The Secretary may, with the approval of the Board of Directors, appoint an Assistant to whom both he and the Secretary of the Council may delegate such of his or their duties as he or they may see fit. This Assistant Secretary shall receive such salary as shall be fixed by the Board of Directors, which shall cover his services both to the Secretary and to the Secretary of the Council.

VI. Treasurer.

The Treasurer shall collect and, under the direction of the Board of Directors, shall disburse all funds of the Institute. He shall keep regular accounts in books belonging to the Institute, which shall be open to any member of the Board of Directors. He shall report in writing at each annual meeting of the Institute and at every meeting of the Board of Directors at which such report shall be called for, the balance of money on hand, and any existing appropriation which may affect the same.

His accounts shall be audited annually by a Committee of three Members or Associates to be appointed by the President at least thirty days prior to the annual meeting in each year, which Committee shall report thereon at such annual meeting.

The Treasurer may, at his discretion, place funds of the Institute, not at any time exceeding $5,000, in a special account in a Bank or Trust Company, subject to the draft of the Assistant Treasurer, and may delegate to the Assistant Treasurer the duty of paying, out of this account, the current expenses of the Institute.

The Treasurer shall be solely responsible to the Institute for all moneys received, whether the same are entrusted to the Assistant Treasurer or not.

VII. Assistant Treasurer.

The Treasurer may appoint, with the approval of the Board of Directors, an Assistant Treasurer, to whom he may delegate the duty of conducting the correspondence incidental to the office of Treasurer, of receiving and depositing in bank to the credit of the Institute all moneys received, and of paying, out of the special account upon which he may be authorized to draw, the necessary expenses of the Institute. The Treasurer may require of him a bond, running to the Treasurer personally, in an amount not exceeding $5,000, the expense of which shall be borne by the Institute.

The Assistant Treasurer shall receive such compensation as shall be fixed by the Board of Directors.

The offices of the Assistant Secretary and of the Assistant Treasurer may, if

so desired by both the Secretary and the Treasurer and approved by the Board of Directors, be united in the same person, who shall then receive the salary of both offices.

The Assistant Treasurer may, with the approval of the Board of Directors, employ such persons as are necessary to constitute a clerical and office force for himself, the Assistant Secretary and the Secretary of the Council, at such salaries as shall be approved by the Board of Directors. He shall, if the offices of Assistant Secretary and Assistant Treasurer be united in the same person, be the immediate superior of all such employees, unless the Secretary of the Council or the Treasurer be present, in which event either of them shall be the superior of all employees, including their respective assistants.

VIII. Standing Committees.

The Standing Committees of the Institute shall be three in number, known respectively as the Finance Committee, the Library Committee and the Committee on Membership.

The Finance Committee and the Library Committee shall each consist of three members of the Board of Directors, and shall be appointed by the President at the first meeting of the Board, after the annual meeting in each year.

The Committee on Membership shall consist of five Members of the Council, and shall be appointed by the President of the Council, at the first meeting of the Council after the first annual meeting in each year.

IX. Finance Committee.

It shall be the duty of the Finance Committee to inquire into and examine the financial condition of the Institute, and to consider ways and means of increasing its revenues and of limiting its expenses. It shall report from time to time to the Board as often as it may deem expedient, and whenever it shall be directed so to do; and the Treasurer shall at all times furnish it with such statements and information as it may desire.

It shall determine the investment of such surplus moneys as shall from time to time accrue to the Institute. It shall, at least once in each year, examine the securities belonging to the Institute in the custody of the Treasurer, and report thereon to the Board.

It may, at any time, examine the books and vouchers of the Treasurer and Assistant Treasurer.

The Treasurer shall not be a member of the Finance Committee, but shall attend the meetings of the same if requested to do so.

X. Library Committee.

The Library Committee shall be the custodian of all books in the Institute Library and of additions thereto; also of all back numbers of the *Transactions* of the Institute. It shall, on the first day of May, of each year, receive from the Secretary of the Council, and receipt for same to him, all the volumes of *Transactions* for the preceding year, not then distributed by said Secretary.

It shall cause to be kept, under the direction of the Assistant Secretary, a catalogue of all books in the Library and an account in ledger form of all volumes of *Transactions* in its custody, in which shall be charged to it all volumes delivered to it, and in which shall be credited all volumes taken from its custody for sale or for any other purpose.

The receipts from the sale of any volume of *Transactions* taken from the custody of the LIBRARY COMMITTEE shall be credited to the LIBRARY COMMITTEE on the books of the Treasurer, and devoted to the general purposes of the Institute.

XI. COMMITTEE ON MEMBERSHIP.

All nominations for Members or Associates of the Institute shall be submitted to and passed upon by the COMMITTEE ON MEMBERSHIP, who shall report thereon to the Council. It shall receive and consider all communications respecting candidates, and shall make diligent inquiry as to the character and qualifications of each one. Its proceedings shall be secret and confidential.

No member of the Committee shall propose any candidate.

XII. ELECTION OF MEMBERS.

After the COMMITTEE ON MEMBERSHIP shall have reported to the Council its conclusions as to the acceptability of each candidate, the Council shall vote upon the same.

Two negative votes of members of the Council present shall prevent the election of any candidate. No person shall be proposed for election to the Institute within one year after his name shall have been rejected by the Council.

XIII. UNITED ENGINEERING SOCIETY.

The Board of Directors shall, at its first meeting after the adoption of these By-Laws, designate three Members or Associates of this Institute to be representatives of this Institute upon the Board of Trustees of the UNITED ENGINEERING SOCIETY, making at the same time provision for the expiration of the terms of office of said representatives, as provided in the By-Laws of the said UNITED ENGINEERING SOCIETY.

At the last meeting of the Board of Directors prior to the first day of each January thereafter, the Board shall designate a Member or Associate of this Institute to be a representative of this Institute upon the Board of Trustees of the said UNITED ENGINEERING SOCIETY for a period of three years beginning at the next ensuing annual meeting of said Society.

At any time when a vacancy shall occur in the representation of this Institute in the Board of Trustees of said Society, by reason of the death, resignation or removal of any such representative therein, the Board of Directors of this Institute shall designate a Member or Associate to fill such unexpired term.

XIV. PUBLICATIONS.

The publications of the Institute shall include a periodical, called the *Bi-Monthly Bulletin* of the American Institute of Mining Engineers, which shall contain reports of proceedings, professional papers, notices, and other matter of interest to members. From the annual dues paid by each Member or Associate, five dollars shall be deducted and applied as a subscription to the *Bi-Monthly Bulletin* for the year covered by such payment.

XV. AMENDMENTS.

These By-Laws may at any time be altered or amended by a vote of two-thirds of the Board of Directors, or by the Members, at a business meeting of the Institute, in the same manner provided for amendments of the Constitution in Article XII. thereof.

ANNUAL MEETING.

The election of officers by vote of the members and associates in person or by proxy at the Annual Meeting, Feb. 20, 1906, resulted as follows:

COUNCIL.

President of the Council.

Robert W. Hunt, ... Chicago, Ill.

(Term expires February, 1907.)

Vice-Presidents of the Council.

Henry M. Howe, ... New York, N. Y.
J. B. Grant, ... Denver, Colo.
James D. Hague, ... New York, N. Y.

(Term expires February, 1908.)

Councilors.

Theodore Dwight, ... New York, N. Y.
Walter Wood, ... Philadelphia, Pa.
Wm. Fleet Robertson, ... Victoria, B. C., Can.

(Term expires February, 1909.)

Secretary of the Council.

R. W. Raymond, ... New York, N. Y.

(Term expires February, 1907.)

Assistant Secretary and Editor (by appointment).

Joseph Struthers, ... New York, N. Y.

DIRECTORS OF THE CORPORATION.

Theodore Dwight, Charles H. Snow, R. W. Raymond.

(Term expires February, 1909.)

ACTS OF THE BOARD OF DIRECTORS.

November 3, 1905, Messrs. Henri Le Chatelier, of Paris, France, and Andrew Carnegie, of New York, N. Y., having been nominated by a large number of members, and recommended by unanimous vote of the Council, were unanimously elected Honorary Members, in recognition of their distinguished services to the arts and professions represented by the Institute.

February 7, 1906, Dr. James Douglas, of New York, N. Y., having been nominated by a large number of leading members, and recommended by unanimous vote of the Council, was unanimously elected an Honorary Member, in recognition of his distinguished services to the practice and the literature of mining and metallurgy, his zealous and loyal devotion to this Institute, his fraternal generosity towards professional colleagues, and his effective advocacy, by public utterance and private example, of the free communication of technical knowledge and experience.

At the same meeting, the resignation of Mr. Theodore Dwight, as Assistant Secretary and Assistant Treasurer of the Institute, offered on account of the conflicting demands of private business, was accepted, and the Secretary was instructed to express to Mr. Dwight the thanks of the Board for his loyal and efficient services, and its regret that he could not continue in that position.

At the same meeting, the Secretary and the Treasurer reported respectively their appointment of Mr. Henry W. B. Howard as Assistant Secretary and Assistant Treasurer, to fill the vacancies created by the resignation of those offices by Mr. Dwight; and the said appointments were confirmed.

February 20, 1906, after the adjournment of the Annual Meeting of the Institute, the Board of Directors elected the following officers for the ensuing year: *President*, James Gayley; *Vice-President*, James Douglas; *Secretary*, R. W. Raymond; *Treasurer*, Frank Lyman.

Financial Statement.

The following statement of receipts and expenditures from Jan. 9 to Dec. 31, 1905, duly audited by Barrow, Wade, Guthrie & Co., certified public accountants, is published by authority of the Board of Directors:

Receipts.

Balance from statement of January, 1905,		$9,401.58
Annual dues,	$29,109.48	
Life memberships,	2,592.49	
Initiation fees,	3,200.28	
Binding of *Transactions*,	2,823.65	
Sale of publications,	2,960.60	
Electrotypes,	72.50	
Miscellaneous receipts,	633.90	
Interest on bonds and deposits,	1,192.88	
		42,585.78
		$51,987.36

Disbursements.

Printing Vol. XXXV. of the *Transactions*,	$4,129.64	
Printing *Bi-Monthly Bulletin* and pamphlet edition of papers,	6,468.35	
Printing circulars and ballots,	349.80	
Printing special editions,	32.25	
Printing new edition "Genesis of Ore-Deposits,"	60.00	
Binding Vol. XXXV. of the *Transactions*,	2,863.36	
Binding miscellaneous volumes,	355.18	
Binding of exchanges,	262.90	
Engraving and electrotyping,	731.31	
Secretary's department, including editorial work,	9,510.00	
Postage, including post-office box rental,	1,755.68	
Stationery,	675.82	
Rent,	2,750.00	
Express and freight charges,	2,557.69	
Treasurer's department, including collection of dues, shipping, etc.,	7,291.34	
Telephone,	280.60	
Telegrams, cables and carfare,	128.11	
Storage of *Transactions*,	180.38	
Office supplies and repairs,	172.93	
Librarian and assistant,	1,360.00	
Refunding payments,	39.99	
Insurance premiums, fire and surety,	277.74	
Collection charges,	35.19	
Extra clerical help,	60.00	
Special stenographers and expenses of meetings,	1,096.64	
Auditing,	125.00	
Office cleaning and sundry expenses,	103.89	
		$43,653.79

Carried over,	$43,653.79
Incorporation expenses,	1,071.75
Office equipments,	143.65
Library additions,	299.21
Balance,	6,818.96
	$51,987.36

NOTE.—It should be borne in mind that by reason of the fact that the incorporation of the Institute was completed Jan. 9, 1905, and the final financial statement of the former Council was brought to that date, the present statement begins with January 9, instead of January 1, as future annual statements will do. It omits, therefore, the receipts and expenditures for the first nine days of 1905, during which period $6,841.62 was received (almost all for dues of 1905), and only $100 was expended (for weekly salaries). In other words, $6,741.62, excess of receipts over expenditures, belonging strictly to 1905, is included in the "Balance from Last Account," at the head of this statement. The receipts for the whole year were, therefore, $49,427.40, and the expenditures $45,278.40, showing a surplus of $4,149.00.

This statement does not include a payment made on account of the Institute to "United Engineering Society," in connection with the Carnegie Engineering Building—the amount of which was supplied from a special fund, raised by voluntary subscription. The full account of this fund will be separately published, at the proper time, by the Committee having it in charge.

REPORT OF THE COUNCIL FOR THE YEAR 1905.

MEETINGS.

Two meetings for the reading of papers, etc., were held during the year, namely, the Eighty-Eighth,* held May 2 to 5, at Washington, D. C., and the Eighty-Ninth, held July 1 to 5, at Victoria, B. C. The proceedings of these meetings, including the record of committees and the presentation of professional papers and a description of the entertainments and excursions connected with these meetings, are to be published in the *Transactions*, Vol. XXXVI. These meetings were highly successful, both professionally, on account of the large number of valuable papers presented, and socially, through the various delightful excursions and entertainments provided for the visiting members and guests.

BI-MONTHLY BULLETIN.

The technical papers which, prior to 1905, had been distributed as separate pamphlets, were issued during this year in the form of a bi-monthly bulletin, under the management of Dr. Joseph Struthers, Assistant Secretary and Editor. In order that the individual technical papers may be filed or otherwise used separately, each one has been paged and wired by itself, the whole collection being held together by a single wire staple, upon the removal of which it will fall apart into individual pamphlets, substantially like those formerly issued. In addition to the technical papers, the *Bulletin* contains announcements of general interest to the members of the Institute, such as lists of accessions to the library, election of new members, etc.

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* Prior to the incorporation of the Institute, the Annual Meeting was simply one of the regular meetings, and was mainly occupied, like the rest, with the reading and discussion of papers. At present, the Annual Meeting is devoted to business only. Consequently, it will not be hereafter numbered serially in the list of meetings.

The six numbers of the *Bulletin* issued during 1905 comprised 1,413 pages of technical papers and discussions, and 115 pages of special announcements, a total of 1,528 pages; thus furnishing to the members more than 500 pages of reading-matter beyond that included in Volume XXXVI. of the *Transactions*.

Membership.

Changes in membership have taken place during the year as follows:—1 honorary member (not previously on the regular list of members), 322 members and 19 associates have been elected; 1 member has been elected an honorary member, and 4 associates have become members; the deaths of 1 honorary member, 34 members and 5 associates have been reported; 46 members and 5 associates have resigned, and 48 members and 1 associate have been dropped from the roll by reason of nonpayment of dues, loss of correct address, etc.* These changes are shown in the accompanying table.

The total membership on Jan. 1, 1906, was 3,884, as compared with 3,680 on Jan. 1, 1905—a net gain for the year of 204 members.

Membership of the American Institute of Mining Engineers, Jan. 1, 1906.

	Honorary Members.	Members.	Associate Members.	Totals.
Membership Dec. 31, 1904	7	3,483	190	3,680
Gains: By Election	1	322	19	342
Change of Status	1	4	...	5
Reinstatement	...	1	...	1
Re-election	...	1	...	1
Losses: By Resignation	...	46	5	51
Dropping	...	48	1	49
Change of Status	...	1	4	5
Death	1	34	5	40
Total gains	2	328	19	349
Total losses	1	129	15	145
Membership Dec. 31, 1905	8	3,682	194	3,884

* Many of these, no doubt, will be reinstated, as has been the case in former years.

The list of deaths reported during the year comprises the following names, the figures in parentheses indicating the year in which the persons named were elected members of the Institute:

Honorary Member.—Dr. Bruno Kerl.

Members and Associates.—Charles Christy Adams (1903), John Wesley Anderson (1898), William Burton Barber (1901), Thomas A. Bennet (1900), John W. Best (1899), Luke W. Bryan (1901), Edward Cooper (1874), W. E. C. Coxe (1874), Edwin S. Daugherty (1905), Isidore Davidov (1900), William Scott de Camp (1875), George H. Eldridge (1889), B. W. Frazier (1871), F. G. Fricke (1879), John Broome Guinn, Jr. (1897), Dana C. Irish (1900), Harry Hugh Lee (1899), William Alfred Lindsay (1902), Robert C. Luther (1884), James MacNaughton (1893), Gordon McLean (1899), Herbert Holmes McNamara (1902), August R. Meyer (1886), William J. Parker, Jr. (1898), Joseph Phillips, Jr. (1895), Charles O. Ripley (1902), Emmerich J. Schmitz (1883), Ludwig Philip Seeger (1900), Jacob Johnston Sperry (1900), Charles B. Squier (1886), Gustavus H. Stoiber (1902), William L. Study (1905), Robert Macnair Wilson Swan (1900), Richard Henry Terhune (1886), Joseph Thiry (1892), John N. Tisdale (1903), Alfred Isham Totten (1897), Walter H. Virgoe (1902), Charles H. Wellman (1890).

[SECRETARY'S NOTE.—Biographical Notices of Bruno Kerl and Benjamin West Frazier were published in Vol. XXXVI. of the *Transactions.* A Biographical Notice of Edward Cooper is given elsewhere in this volume. Concerning the remaining names in the above list, such data as the Secretary was able to obtain appeared in *Bi-Monthly Bulletin*, No. 9, May, 1906, under the title, "Biographical Notices of 1905."—R. W. R.]

MEMBERSHIP.

The following list comprises the names of those persons elected as Members, who duly accepted election during the year 1906. The following marks are used to designate the different classes of membership: Heavy-faced type, Honorary Member; **, Life Member; *, Member; ††, Life Associate Member; †, Associate Member.

*Arthur K. Adams,	Spencer, Mass.
**John C. Adams,	Butte, Mont.
*Ralph E. Adams,	Kellogg, Idaho.
*William Adams,	Chihuahua, Mex.
*William Affleck,	Ellsworth, Pa.
*Frederick E. Allen,	Concepcion del Oro, Zacatecas, Mex.
*Ulysses S. S. Arentz,	Salt Lake City, Utah.
*Austin J. R. Atkin,	Johannesburg, Transvaal, So. Africa.
*Robert C. Atkinson,	Johannesburg, Transvaal, So. Africa.
*W. J. Atkinson,	Winnipeg, Can.
*Percy A. Babb,	Mexico City, Mex.
*George A. Baird,	Chicago, Ill.
*William S. Baldwin,	Villa Grove, Colo.
*John H. Bartlett,	New Market, Tenn.
*John R. Bartlett,	Butte, Mont.
*Owen S. Batchelor,	Kamloops, B. C., Can.
*Clarence A. Beall,	Idaho Springs, Colo.
*James T. Beard,	Scranton, Pa.
*John H. Bennetts,	Gympie, Queensland, Australia.
*Richard G. Bevington,	Johannesburg, Transvaal, So. Africa.
*Stephen Birch,	New York, N. Y.
*Henry D. Boddington,	El Paso, Tex.
*Frank K. Boggs,	Temosachic, Chihuahua, Mex.
*Louis W. Bond,	Goldfield, Nev.
*Henry Bossuet,	El Oro, Mexico, Mex.
*Pierre Bouéry,	Weaverville, Cal.
*Alexander J. Bourdariat,	Paris, France.
*Charles H. Bowman,	Butte, Mont.
*Horace Boyd,	Hokendauqua, Pa.
*Oliver U. Bradley,	Hornitos, Cal.
*Norman B. Braly,	Butte, Mont.
*George Bright,	Pottsville, Pa.
**Robert S. Brooks,	Ocampo, Chihuahua, Mex.
*Edward W. Brown,	New York, N. Y.
*George P. Brown,	San Diego, Cal.
*Harvey S. Brown,	De Lamar, Idaho.

*Josephus J. Brown, Jr., ... Granby, Mo.
*Charles A. Buck, ... South Bethlehem, Pa.
*Peter A. Busch, ... Rhyolite, Nev.
*John S. Butler, ... Guanajuato, Guanajuato, Mex.
*Emmett A. Byler, ... Goldfield, Nev.
*John Cadman, ... Port of Spain, Trinidad, W. I.
*Milton A. Caine, ... Isabella, Tenn.
**John M. Callow, ... Salt Lake City, Utah.
*J. Edouard Capdeville, ... Paris, France.
*Albert E. Carlton, ... Cripple Creek, Colo.
*Ellard W. Carson, ... Cambria, Cal.
*Albert H. Case, ... Ophir, Utah.
*Thomas S. Chalmers, ... Chicago, Ill.
*Clarence C. Chase, ... Temosachic, Chihuahua, Mex.
*Frank O. Clements, ... Dayton, O.
*James C. Climo, ... Mackay, Idaho.
*Robert W. Coats, ... Sharpsville, Pa.
*Frank F. Colcord, ... Maurer, N. J.
*Edward B. Cook, ... Pottstown, Pa.
*James Cook, ... Goldfield, Nev.
**Hugh L. Cooper, ... New York, N. Y.
*Reginald W. G. Corfield, ... Swansea, England.
*Robert R. Cormack, ... Rossland, B. C., Can.
*Paul S. Couldrey, ... Rossland, B. C., Can.
*Frederick Cowans, ... Ocampo, Chihuahua, Mex.
*Edward O. Cross, ... Fostoria, Ohio.
*Milton T. Culbert, ... Cobalt, Ont., Can.
*John M. Currie, ... Day Dawn, Western Australia.
*O. Robert Dahl, ... Seattle, Wash.
*Henry G. Dalton, ... Cleveland, O.
*George G. Damon, ... Iola, Kan.
*William A. T. Davies, ... Bulong, Western Australia.
*Daniel Davis, ... Alburtis, Pa.
*Thomas B. Davis, Jr., ... New York, N. Y.
†Charles F. De Armond, ... Jualin, Alaska.
*Arthur J. Debenham, ... Beaconsfield, Tasmania.
*George H. Deike, ... Mosgrove, Pa.
*Harry S. Derby, ... Chicago, Ill.
*William G. Devereux, ... Glenwood Springs, Colo.
*Alfred J. Diescher, ... Pittsburg, Pa.
*William P. J. Dinsmoor, ... Denver, Colo.
*Leon Dominian, ... Bonanza, Zacatecas, Mex.
*W. H. Donner, ... Pittsburg, Pa.
*Ross E. Douglass, ... Bulawayo, Rhodesia, So. Africa.
*Walford R. Dowling, ... Germiston, Transvaal, So. Africa.
*Carl H. Draper, ... Etzatlan, Jalisco, Mex.
*Charles V. Drew, ... New York, N. Y.
*Will W. Duffield, ... New York, N. Y.
*Damon D. Dunken, ... Joplin, Mo.
*Drew H. Dunn, ... Pasadena, Cal.

*Harry D. Easton, ... Hartshorne, Ind,
*Albert L. Eaton, ... Chihuahua, Mex.
*Horace W. Edmondson, ... Chihuahua, Mex.
**Victor E. Edwards, ... Worcester, Mass.
*Mark Ehle, Jr., ... Rapid City, S. D.
*George R. Elder, ... Easton, Pa.
*Albert S. Ellam, ... London, England.
†Robert P. Elliott, ... Oaxaca, Oaxaca, Mex.
*Stuart R. Elliott, ... Negaunee, Mich.
†William H. Falding, ... Rossland, B. C., Can.
*Benjamin L. Farrar, ... Chihuahua, Mex.
*Sydney Fawns, ... London, England.
*Donald Ferguson, ... Goldfield, Nev.
**Henry G. Ferguson, ... Ishpeming, Mich.
*John J. C. Fernau, ... Nelson, B. C., Can.
*Morrison Fetzer, ... Chihuahua, Mex.
*William N. Fink, ... Niblack, Alaska.
*Francis S. Foote, Jr., ... Montclair, N. J.
*Herbert W. Fox, ... Colorado Springs, Colo.
*Rudolf Franke, ... Eisleben, Germany.
*Alexander J. Fraser, ... Rhyolite, Nev.
*Colin Fraser, ... Wellington, New Zealand.
*Lester D. Frink, ... Butte, Mont.
*Arthur W. Geiger, ... Ketchikan P. O., Alaska.
*Jerome R. George, ... Worcester, Mass.
*Walter G. Gleeson, ... Granite, Oregon.
*Edgar M. Gleim, ... Portland, Ore.
*Robert H. Goodwin, ... Seaford, Sussex, England.
*Stanley N. Graham, ... Guanajuato, Mex.
*J. Sharshall Grasty, ... Staunton, Va.
*Louis C. Graton, ... Washington, D. C.
*Fitz Roy N. Griffin, ... Chicago, Ill.
*C. Godfrey Gunther, ... Brooklyn, N. Y.
†Edward A. Guntly, ... Cripple Creek, Colo.
Robert A. Hadfield, ... Sheffield, England.
*William Hague, ... New York, N. Y.
*Frank R. Hall, ... Telluride, Colo.
*Alexander Hamilton, ... Goldfield, Nev.
*Edward L. Hammond, ... Asientos, Aguascalientes, Mex.
*Henry H. S. Handy, ... Syracuse, N. Y.
*Nils V. Hansell, ... New York, N. Y.
*Leonard Harrison, ... Wellsboro, Pa.
*Philip E. Hart, ... Nelson, B. C., Can.
*James C. Hartness, ... Tombstone, Ariz.
*Elmer C. Heck, ... Lathrop, Mo.
*Edwin T. Henderson, ... Broken Hill, N. S. W., Australia.
*Hyman Herman, ... Mt. Bischoff, Tasmania.
*Earl E. Hewitt, ... Dunglen, Ohio.
*Ernest Hibbert, ... Akmolinsk, Siberia.
*John I. Hoffmann, ... Johannesburg, Transvaal, So. Africa.
*William Howard, ... El Oro, Mexico, Mex.

*Harry J. Hubbard, ... Ocampo, Chihuahua, Mex.
*John P. Hutchins, ... New York, N. Y.
*Charles A. Hyder, ... Moctezuma, Sonora, Mex.
**Stephen A. Ionides, ... Hove, Sussex, England.
*Thomas A. Irvin, ... Denver, Colo.
*George D. James, ... Reno, Nev.
*Albert B. Jessup, ... Wilkes-Barre, Pa.
*David John, ... Haileybury, Ontario, Can.
*Paul Johnson, ... Hadley, Ketchikan P. O., Alaska.
*William Jones, ... Watford, Herts, England.
*George L. Kaeding, ... Goldfield, Nev.
*Daniel B. Kane, ... Cerro de Pasco, Peru, So. America.
*Yoichi Katsura, ... Tokio, Japan.
*Frederic Keffer, ... Greenwood, B. C., Can.
**Frank A. Keith, ... Tonopah, Nev.
*Henry S. Kenney, ... Johannesburg, Transvaal, So. Africa.
*Oliver King, ... Johannesburg, Transvaal, So. Africa.
*Tom Cobb King, ... New York, N. Y.
*A. J. Klamt, ... Tonopah, Nev.
*Jintaro Kojima, ... Shimotsuke, Japan.
*Robert B. Lamb, ... Hedley, B. C., Can.
*William F. Lamoreaux, ... Isabella, Tenn.
*Thomas Lancaster, ... New York, N. Y.
*Albert G. Langley, ... Victoria, B. C., Can.
†Enrique E. Laroza, ... Brussels, Belgium.
*Wallace Lee, ... Ures, Sonora, Mex.
*Frank M. Leland, ... Mackay, Idaho.
*James B. Lewis, ... Lottah, Tasmania.
*Thomas L. Livermore, Jr., ... San Fernando, Durango, Mex.
*John R. Lucas, ... Philipsburg, Mont.
*Percy K. Lucke, ... London, England.
*Charles J. Lyser, ... San Francisco, Cal.
*Welbert G. McBride, ... Bisbee, Ariz.
*Walter E. McCourt, ... Ithaca, N. Y.
*Wallace D. McDougall, ... London, England.
*William L. McLaughlin, ... Deadwood, So. Dak.
*Arthur C. Macdonald, ... Antofagasta, Chile, S. A.
*Henry S. Mackay, ... Santa Ana, Sonora, Mex.
*Edmund von Maltitz, ... South Chicago, Ill.
*Frank A. Marriott, ... Orange, N. S. W., Australia.
*James J. Martin, ... Barranquilla, Colombia, So. America.
*C. G. Memminger, ... Lakeland, Fla.
*Wallace W. Merriam, ... Estacion Terrazas, Chihuahua, Mex.
*William G. Metzger, ... Fairmont, W. Va.
*John B. Miles, ... Philadelphia, Pa.
*Charles N. Milne, ... Algoma, Ontario, Can.
*Charles T. Mitchell, ... Grand Forks, B. C., Can.
*Henry H. Mitchell, ... South Bethlehem, Pa.
*William E. C. Mitchell, ... Johannesburg, Transvaal, So. Africa.
**Warden A. Moller, ... Tientsin, China.
*Redick R. Moore, ... Oaxaca, Oaxaca, Mex.

*Donald R. Morgan, ... Monterey, N. L., Mex.
*Paul B. Morgan, ... Worcester, Mass.
*George B. Morris, ... Concord, N. C.
*Harry T. Morris, ... Bethlehem, Pa.
*Bradish P. Morse, ... Denver, Colo.
*Bryan K. Morse, ... Cananea, Sonora, Mex.
*Erle D. Morton, ... Ely, Nev.
*John A. W. Murdoch, ... Cerro de Pasco, Peru, So. America.
*Francis J. Murphy, ... La Cienequita, Sonora, Mex.
*Winthrop C. Neilson, ... Rome, Ga.
*John L. Nelson, ... Prescott, Ariz.
*Bruno Newman, ... Asientos, Aguascalientes, Mex.
*Paul Nicholas, ... Springfield, Mo.
*Burton B. Nieding, ... Niblack, Alaska.
*Frederick W. Nobs, ... Minillas, Zacatecas, Mex.
*Edwin G. N. North, ... Akmolinsk, Siberia.
*Joseph F. O'Byrne, ... Dahlonega, Ga.
*William A. Paine, ... Boston, Mass.
*Cyril E. Parsons, ... Lomagundi, Rhodesia, So. Africa.
*Peter C. Patterson, ... Pittsburg, Pa.
*Frank Peterson, ... New York, N. Y.
†Newton W. Pilger, ... Ruby, Mont.
*John R. Pill, ... Carbon Hill, Ala.
*Ernest Y. Pomeroy, ... Kofa, Ariz.
†Charles F. Potter, ... Denver, Colo.
*Russell Prentice, ... San Francisco, Cal.
*Harry B. Price, ... Mexico City, Mex.
*Harrie B. Pulsifer, ... Lebanon, N. H.
*Dana G. Putnam, ... La Union, Salvador, Central America.
†Frederick B. Reece, ... Kellogg, Idaho.
*Joseph W. Reid, ... Yellville, Ark.
*Herman Reischke, ... Reveille, Nev.
*Lucien S. Robe, ... Fairbanks, Alaska.
*George H. Robinson, Jr., Minas Pena del Hierro, Provincia de Huelva, Spain.
*Charles G. Roebling, ... Trenton, N. J.
*Ferdinand W. Roebling, Jr., ... Trenton, N. J.
*Diedrich P. Rohlfing, ... Salt Lake City, Utah.
*Lewis P. Ross, ... Johnson City, Tenn.
*Milton L. Rubel, ... Mascota, Jalisco, Mex.
*J. Arthur Rule, ... Torreon, Coahuila, Mex.
*William R. Rumbold, ... Oruro, Bolivia, So. America.
*H. Yelverton Russel, ... Ouray, Colo.
*Earl F. Salisbury, ... Alamo, Nueva Leon, Mex.
*William L. Saunders, ... New York, N. Y.
*John A. Savage, ... Pittsburg, Pa.
*Arthur H. Sawyer, ... Greenland, Mich.
*Charles W. Saxman, Jr., ... Latrobe, Pa.
*Ehrich J. Schrader, ... St. Paul, Minn.
*Harold Sharpley, ... Kanowna, Western Australia.
*Edward L. Shera, ... Ocampo, Zacatecas, Mex.
†James C. Simpson, Jr., ... Edinburgh, Scotland.

*George O. Smart, ... Germiston, Transvaal, So. Africa.
*Alexander H. Smith, ... Oaxaca, Mex.
*Walter F. Smith, ... Philadelphia, Pa.
*Albert E. Smyser, ... Steubenville, O.
**Walter O. Snelling, ... Washington, D. C.
*William C. Snyder, ... Snow Shoe, Pa.
*O. H. Sonne, ... Yerington, Nev.
*Howard Spangler, ... Victor, Colo.
*Persifor G. Spilsbury, ... Cananea, Sonora, Mex.
John E. Stead, ... Sheffield, England.
*Horace J. Stevens, ... Houghton, Mich.
*Norman C. Stines, ... Carrville, Cal.
*James Stirling, ... Hollywood, Cal.
*Arthur H. Stockdale, ... Cayuca de Catalan Guerrero, Mex.
**Thomas W. P. Storey, ... Mostyn, N. W. England.
*Hallyburton T. Stretton, ... London, England.
*Harold Sturges, ... Mexico City, Mex.
*John M. Sully, ... Hanover, N. M.
*Maurice W. Summerhayes, ... Beckenham, Kent, England.
*Harry L. Swain, ... Mexico City, Mex.
*Morris G. Talcott, ... Clifton, Ariz.
*Oscar Textor, ... Cleveland, O.
*Henry T. Thomas, ... Champion Reefs, Mysore State, India.
*Hubert S. Thomas, ... Wellfield, Llanelly, England.
*James E. Thomas, ... Krugersdorp, Transvaal, So. Africa.
*John W. Thomas, ... Island Park, Pa.
*Joseph F. Thorn, ... Chittabalbie, Korea.
*Colin Timmons, ... Taxco, Guerrero, Mex.
*Cyrus F. Tolman, Jr., ... Tucson, Ariz.
*John D. Tolman, Jr., ... Aspen, Colo.
*Carl Tombo, ... New York, N. Y.
*William W. Trowbridge, ... Hacolula, Oaxaca, Mex.
*Scott Turner, ... Lansing, Mich.
*William D. Verschoyle, ... Ballisodare, Ireland.
*Juan D. Villarello, ... Mexico City, Mex.
*Joseph C. Vivian, ... Rio Tinto Mines, Provincia de Huelva, Spain.
*Felix A. Vogel, ... Florence, Wis.
†Ludwig Vogelstein, ... New York, N. Y.
†George F. Vollmer, ... London, England.
*Arthur H. Walker, ... New York, N. Y.
†Edward J. Walsh, ... St. Louis, Mo.
*Isaac S. Weaver, ... Bellevue, Idaho.
*Ray H. Webb, ... Aguascalientes, Mex.
*George E. Webber, ... Johannesburg, Transvaal, So. Africa.
*George E. Werner, ... Los Angeles, Cal.
*Harry K. Wheeler, ... Ely, Nev.
*Edmond A. While, ... Mt. Morgan, Australia.
*Jeffries White, ... Ocotlan, Oaxaca, Mex.
*Charles Wilkinson, ... Whitehaven, England.
†Robert H. Willets, ... Roslyn, L. I., N. Y.
**Alfred W. G. Wilson, ... Montreal, Can.

*John B. Wilson,	Cerro de Pasco, Peru, So. America.
*Odell Wilson,	Salmon, Idaho.
*John L. Winter,	San Francisco, Cal.
*Mark A. Wolff,	Wei Hai Wei, China.
*Stuart Wood,	Philadelphia, Pa.
*Dwight E. Woodridge,	Duluth, Minn.
*Francisco J. Woods,	Oaxaca, Mex.
*Harold J. Wright,	Hillgrove, N. S. W., Australia.
*Lars Yungstrom,	Falun, Sweden.
*Martin B. Zerener,	New York, N. Y.

Deaths.

The following list comprises the names of members whose deaths have been reported to the Secretary of the Institute during the year 1906:

Date of Election.	Name.	Date of Decease.
1882.	*Abbott, Arthur Vaughan,	December 1, 1906.
1889.	*Akers, William Anderson,	June 17, 1906.
1905.	*Allen, R. Scott,	February —, 1906.
1900.	*Arlett, George H.,	March 19, 1906.
1883.	**Austin, T. S.,	August 23, 1906.
1902.	*Batchelor, William Tittley,	September 27, 1906.
1897.	**Bell, Charles Lowthian,	February 8, 1906.
1892.	*Bensusan, Edgar Vallentine,	May 8, 1906.
1896.	*Breisch, Ernest E.,	February 14, 1906.
1895.	*Brown, Horace F.,	April 15, 1906.
1876.	*Burden, James A.,	September 23, 1906.
1880.	†Coxe, Alexander B.,	January 23, 1906.
1879.	*Crocker, George A.,	October 20, 1906.
1904.	**Fraser, John H.,	April 1, 1906.
1892.	*Gibson, Robert,	April 18, 1906.
1900.	*Hart, R. G.,	November 14, 1906.
1899.	*Jolly, Alexander W.,	————————.
1899.	*Keener, George L.,	August 13, 1906.
1893.	*Lanning, John G.,	July 18, 1906.
1882.	*Lennig, Nicholas,	January 24, 1906.
1905.	*Mandell, Frank C.,	December 14, 1906.
1895.	*Miller, Edmund Howd,	November 8, 1906.
1903.	**Morris, John Fossbrook,	December 23, 1905.
1893.	*Odling, Francis James,	September 10, 1906.
1897.	*Orr, William,	November 20, 1905.
1893.	*Painter, William,	July 15, 1906.
1896.	*Pearce, Stanley H.,	July 10, 1906.
1900.	*Poole, Herman,	February 6, 1906.
1898.	*Ramos, Ricardo G.,	————————.
1904.	*Rising, Arthur F.,	October 10, 1906.
1886.	*Robinson, George H.,	July 3, 1906.

* Member. † Associate. ** Life Member.

Date of Election.	Name.	Date of Decease.
1905.	*Sayles, Albert W.,	January —, 1906.
1888.	*Seddon, Richard J.,	June 10, 1906.
1887.	*Simpson, James C.,	September 9, 1906.
1889.	*Sperry, Francis L.,	April 17, 1906.
1877.	†Stanton, John,	February 23, 1906.
1877.	**Stoiber, Edward G.,	April 21, 1906.
1871.	*Thomas, Samuel,	February 21, 1906.
1900.	*Toll, Abel Hyde,	September 13, 1906.
1896.	**Wetherill, John Price,	November 9, 1906.
1903.	*Williams, Harvey Ladew	August 4, 1905.
1901.	*Wrinkle, Lawrence F. J.,	——— —, 1904.

* Member. † Associate. ** Life Member.

Proceedings of the Ninetieth Meeting, Bethlehem, Pa., February, 1906.

COMMITTEES.

General Local Committee.—John Fritz, *Chairman;* A. B. de Saulles, *Vice-Chairman;* R. M. Bird, *Treasurer;* Henry S. Drinker, *Secretary;* Robert H. Sayre, E. P. Wilbur, Charles M. Dodson, Abraham S. Schropp, E. M. McIlvain, Archibald Johnston, A. B. Fichter, G. G. Convers, W. A. Wilbur, Garrett B. Linderman, Frederick Conlin, George Pettinos, Albert Brodhead, Garrett L. Hoppes, M. B. Cutter, H. J. Seaman, C. A. Matcham, B. F. Fackenthal, Jr., A. L. J. Queneau, Geo. R. Elder, Eldridge Wilbur, William H. Chandler, Mansfield Merriman, Joseph F. Klein, William S. Franklin, William Esty, Joseph W. Richards, Howard Eckfeldt, John D. Irving.

Transportation Committee.—H. J. Seaman, C. A. Matcham, B F. Fackenthal, Jr., A. B. Fichter, A. L. J. Queneau, Henry S. Drinker.

Hotel Committee.—Garrett B. Linderman, Charles M. Dodson, W. A. Wilbur, Garrett L. Hoppes, Albert Brodhead.

Steering Committee.—William S. Franklin, William Esty, John D. Irving, R. M. Bird, the members of the Tau Beta Pi Society of Lehigh University.

Committee on Meetings.—Henry S. Drinker, Joseph W. Richards, Howard Eckfeldt.

Entertainment Committee.—W. A. Wilbur, E. M. McIlvain, Archibald Johnston, A. B. de Saulles, G. G. Convers, A. L. J. Queneau, Garrett B. Linderman, Frederick Conlin, George Pettinos, Garrett L. Hoppes, Henry S. Drinker, William S. Franklin, Eldridge Wilbur, Robert M. Bird, *Secretary.*

The opening session was held at 8.30 p.m., Wednesday, Feb. 21, 1906, in Gymnasium Hall of Lehigh University, which had been tastefully decorated with flags and bunting.

Dr. Henry S. Drinker, President of Lehigh University, and Secretary of the General Local Committee, called the meeting to order, and welcomed the Institute to the Bethlehems, where its first meeting, after its organization in May, 1871, at Wilkes-Barre, was held in August, 1871.*

* Dr. Drinker was one of the 22 members—of whom only four survive—who organized the Institute at Wilkes-Barre; and at the Bethlehem meeting which followed, he presented a paper on The Works and Mines of the Lehigh Zinc Co. As President of Lehigh University, he succeeded, in 1905, the late Dr. Thomas M. Drown, another of the "original twenty-two." In his address of welcome, Dr. Drinker emphasized the intimate relations between the Institute and Lehigh University, established through the active membership of many Lehigh professors and graduates.

President James Gayley replied in a brief address, expressing the thanks of the Institute for the cordial welcome thus extended to it, and the pleasure with which so many of its members, especially of those who had participated in its early history, were now gathered once more in the Lehigh valley, which was the cradle of its infancy. In addition to what had been said already by President Drinker, Mr. Gayley recalled the names of many Lehigh valley men, who had been pioneers of the new industrial era, as well as potent factors in the success of the Institute. In this connection, he called attention to the significant fact that the members who joined the Institute during the first year of its existence came largely from the Lehigh valley, or were connected with the industries which it represented, and that of the names added during that period to the membership of the Institute, a surprisingly large number have become famous as scientific investigators and instructors, or as "captains of industry." In other words, the men who founded the Institute, or rallied to its support during the first year which followed, were men of first-class ability and promise, as attested by their achievements and present fame.

The Secretary announced the receipt of telegraphic notice of the death of Samuel Thomas, Catasauqua, Pa., an old and highly esteemed member of the Institute, and son of David Thomas, its first President.

The following paper was presented by the Secretary, in the absence of the author:

Biographical Notice of George H. Eldridge, by S. F. Emmons, Washington, D. C.

The following papers were presented in oral abstract by the author:

Biographical Notice of Edward Cooper, by R. W. Raymond, New York, N. Y.

Biographical Notice of Alexander B. Coxe, by R. W. Raymond, New York, N. Y.

Mr. Howard W. DuBois, of Philadelphia, Pa., gave an admirable and enjoyable exhibition of colored pictures of mountain scenery in British Columbia, Alaska, and the Yukon Territory, which was especially interesting to those who had attended the British Columbia Excursion of the last meeting of

the Institute, and had traveled over much of the country described by Mr. DuBois.

The second session was held at 10 a.m., February 22, in the Physical Lecture-Room of Lehigh University, President Gayley in the chair.

The following papers were presented by the authors, and discussed:

Notes on the Gayley Dry-Air Blast-Process, by Carl A. Meissner, New York, N. Y. (Discussed by Messrs. H. M. Howe, Joseph W. Richards, Frank Firmstone, R. W. Raymond and James Gayley.)

Piping in Steel Ingots, by N. Lilienberg, Philadelphia, Pa.

The Present Condition of the Metallurgy of Aluminum, by Joseph W. Richards, Lehigh University, South Bethlehem, Pa.*

The third session was held at the same place, at 2.30 p.m., February 22. The following paper was read by the Secretary, in the absence of the author:

A Novel Method of Mining Kaolin, by Albert R. Ledoux, New York, N. Y.

The following papers were presented by their authors, and discussed:

A Reconnaissance for the Platinum Metals in British Columbia (illustrated with lantern-views), by Howard W. DuBois, Philadelphia, Pa.*

The Secondary Enrichment of Copper-Iron Sulphides, by Thomas T. Read, New York, N. Y.

The following papers were presented in printed pamphlets:

An Old Specimen of American Spiegeleisen, by Frank Firmstone, Easton, Pa.

Bibliography of Coal-Washing, by S. S. Wyer, Columbus, O.

The Beard-Mackie Sight-Indicator for the Measurement of Marsh-Gas in Collieries, by M. H. Harrington, Philadelphia, Pa.

Fine Grinding of Ore by Tube-Mills, and Cyaniding at El Oro, Mexico, by Gelasio Caetani, Rome, Italy, and E. Burt, El Oro, Mex.

Crushing-Tests of the Diamonds Used in Drilling, by Alexander N. Mitinsky, St. Petersburg, Russia.

* Not furnished for publication.

The Mining, Preparation and Smelting of Virginia Zinc-Ores, by Thomas L. Watson, Blacksburg, Va.

The Ancient Copper-Mines of Lake Superior, by Alvinus Brown Wood, Detroit, Mich.

The Relative Merits of Large and Small Drilling-Machines in Development-Work, by Frederick T. Williams, Victor, Colo.

A Reference-Scheme for Mine-Workings, by Wilbur E. Sanders, Helena, Mont.

Cost-Accounts of Gold-Mining Operations, by T. H. Sheldon, Victor, Colo.

Screens for Sizing, by Ernest A. Hersam, Berkeley, Cal.

Notes on Southern Nevada and Inyo County, California, by Harry H. Taft, Denver, Colo.

The Geology and Petrography of the Goldfield Mining-District, Nevada, by John B. Hastings, Denver, Colo., and Charles P. Berkey, New York, N. Y.

The Mojave Mining-District of California, by Charles E. W. Bateson, New York, N. Y.

Discussion of the paper of James Gayley, on The Application of Dry-Air Blast to the Manufacture of Iron, by Edward D. Campbell, Ann Arbor, Mich.*

Discussion of the paper of H. R. Hall, on The Use of High Percentages of Fine Ore in a Charcoal Blast-Furnace, by R. H. Sweetser, Sault Ste. Marie, Ontario, Can.†

Discussion of the paper of H. H. Campbell, on The Influence of Carbon, Phosphorus, Manganese and Sulphur on the Tensile Strength of Open-Hearth Steel, by Mansfield Merriman, South Bethlehem, Pa.‡

Discussion of the paper of Jüptner von Jonstorff and others on Comparison of Methods for the Determination of Carbon and Phosphorus in Steel, by Clemens C. Jones, Richmond, Va., and R. W. Raymond, New York, N. Y.§

Discussion of the paper of James P. Roe, on The Manufacture and Characteristics of Wrought-Iron, by Taylor Allderdice, Pittsburg, Pa.||

Discussion of the paper of M. R. Campbell, on The Commercial Value of Coal-Mine Sampling, by A. Bement, Chicago, Ill.¶

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* *Trans.*, xxxvi., 765. † *Trans.*, xxxvi., 835. ‡ *Trans.*, xxxvi., 803.
§ *Trans.*, xxxvi., 741. || *Trans.*, xxxvi., 823. ¶ *Trans.*, xxxvi., 834.

Discussion of the paper of M. R. Campbell, on The Classification of Coals, by Dr. Persifor Frazer, Philadelphia, Pa.*

Reply to Mr. Rose's Discussion of the paper by Hofman and Magnuson, on The Effect of Silver on the Chlorination and Bromination of Gold, by H. O. Hofman, Boston, Mass.†

Discussion of the paper of Messrs. Gibb and Philp, on The Constitution of Mattes Produced in Copper-Smelting," by Edward Keller, Baltimore, Md.‡

The following papers were read by title, for future publication:

Gold-Dredging in the Urals, with Notes on Dredging in Siberia, by William H. Shockley, New York, N. Y.

Notes on the Roumanian Oil-Fields, by P. Charteris A. Stewart, Cairo, Egypt.

Entertainments and Excursions.

An account of the entertainments and excursions in which the members of the Institute and their guests participated was published in *Bi-Monthly Bulletin*, No. 9, May, 1906, pp. 501 to 504.

Members and Guests Registered.

The following list, which is doubtless incomplete, comprises the names of members and guests registered at the headquarters:

E. O. C. Acker, Bethlehem, Pa.
William Afflich, Ellsworth, Pa.
Edgar A. Ashcroft, London, England.
W. S. Ayres, Hazleton, Pa.
Mrs. W. S. Ayres, Hazleton, Pa.
F. E. Bachman, Port Henry, N. Y.
Mrs. F. E. Bachman, Port Henry, N. Y.
H. Foster Bain, Urbana, Ill.
David Baker, Philadelphia, Pa.
Robert M. Bird, South Bethlehem, Pa.
Cyrus Borgner, Philadelphia, Pa.
Mrs. Cyrus Borgner, Philadelphia, Pa.
Miss Borgner, Philadelphia, Pa.
Horace Boyd, Hokendauqua, Pa.
John Boyer, Bethlehem, Pa.
Albert Brodhead, Bethlehem, Pa.
Robert E. Brooke, Birdsboro, Pa.
Otto C. Burkhart, Bethlehem, Pa.
R. C. Carpenter, Ithaca, N. Y.
Charles Catlett, Staunton, Va.
W. H. Chandler, South Bethlehem, Pa.
S. H. Chauvenet, Sheridan, Pa.
S. P. Christie, High Bridge, N. J.
Mrs. S. P. Christie, High Bridge, N. J.
E. T. Clymer, Philadelphia, Pa.
F. H. Clymer, Reading, Pa.
L. S. Clymer, Riegelsville, Pa.
Mrs. L. S. Clymer, Riegelsville, Pa.
W. B. Cogswell, Syracuse, N. Y.
Mrs. W. B. Cogswell, Syracuse, N. Y.
Frederick Conlin, Bethlehem, Pa.
G. G. Converse, South Bethlehem, Pa.
C. B. Cook, Pottstown, Pa.
Edgar S. Cook, Pottstown, Pa.
Mrs. Edgar S. Cook, Pottstown, Pa.
Edward B. Cook, Pottstown, Pa.
M. B. Cutter, Bethlehem, Pa.
Daniel Davis, Alburtis, Pa.
A. B. de Saulles, South Bethlehem, Pa.
Mrs. A. B. de Saulles, So. Bethlehem, Pa.

* *Trans.*, xxxvi., 825. † *Trans.*, xxxvi., 802. ‡ *Trans.*, xxxvi., 837.

Mrs. C. M. de Saulles, So. Bethlehem, Pa.
John H. de Saulles, New York, N. Y.
W. A. Doble, San Francisco, Cal.
Charles M. Dodson, Bethlehem, Pa.
P. J. Donohue, Salt Lake City, Utah.
B. I. Drake, New York, N. Y.
H. S. Drinker, South Bethlehem, Pa.
Howard W. DuBois, Philadelphia, Pa.
Mrs. H. W. DuBois, Philadelphia, Pa.
Howard Eckfeldt, South Bethlehem, Pa.
Mrs. H. Eckfeldt, South Bethlehem, Pa.
George R. Elder, Ellsworth, Pa.
Lincoln Ellsworth, Bethlehem, Pa.
William Esty, Montclair, N. J.
Ernst F. Eurich, Bethlehem, Pa.
Harold G. Eynon, Philadelphia, Pa.
Thomas M. Eynon, Philadelphia, Pa.
B. F. Fackenthal, Jr., Riegelsville, Pa.
Mrs. B. F. Fackenthal, Jr., Riegelsville.
A. B. Fichter, Bethlehem, Pa.
Frank Firmstone, Easton, Pa.
W. S. Franklin, Bethlehem, Pa.
John Fritz, Bethlehem, Pa.
J. W. Fuller, Jr., Catasauqua, Pa.
Mrs. J. W. Fuller, Jr., Catasauqua, Pa.
E. W. Furst, Cleveland, O.
James Gayley, New York, N. Y.
Fritz Gleim, Portoferraio, Italy.
G. McM. Godley, New York, N. Y.
H. H. Godshall, Lonsdale, Pa.
Charles Goodman, New York, N. Y.
R. W. Gordon, Rhyolite, Nev.
Charles B. Green, Easton, Pa.
Frank Haas, Fairmount, W. Va.
N. V. Hansell, Lyon Mt., N. Y.
F. E. Haverstick, Philadelphia, Pa.
C. W. Hayes, Washington, D. C.
Homer W. Hendricks, Bethlehem, Pa.
Henry H. Hendshaw, Albany, N. Y.
H. V. Hesse, Fairmount, W. Va.
L. Holbrook, New York, N. Y.
Mrs. L. Holbrook, New York, N. Y.
J. H. Holderness, South Bethlehem, Pa.
Garret L. Hoppes, Bethlehem, Pa.
Henry M. Howe, New York, N. Y.
G. A. Howells, New York, N. Y.
Charles E. Hulick, Hellertown, Pa.
C. S. Hurd, New York, N. Y.
Edward S. Hutchinson, Newtown, Pa.
Mrs. E. S. Hutchinson, Newtown, Pa.
Walter R. Ingalls, New York, N. Y.
John D. Irving, South Bethlehem, Pa.
Woolsey McA. Johnson, Iola, Kan.
Archibald Johnston, Bethlehem, Pa.
W. J. Johnston, New York, N. Y.
Mrs. W. J. Johnston, New York, N. Y.
J. H. Jowett, New York, N. Y.
Edward K. Judd, New York, N. Y.
James F. Kemp, New York, N. Y.
J. S. Kennedy, Stanhope, N. J.
Paul S. King, New York, N. Y.
Charles Kirchhoff, New York, N. Y.
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G. B. Linderman, South Bethlehem, Pa.
Stuart Lindesley, Orange, N. J.
Mrs. Stuart Lindesley, Orange, N. J.
C. P. Linville, State College, Pa.
L. P. Lohne, Cleveland, O.
E. J. McCaustland, Ithaca, N. Y.
E. M. McIlvain, South Bethlehem, Pa.
Mrs. E. M. McIlvain, So. Bethlehem, Pa.
William R. McIlvain, Reading, Pa.
Mrs. William R. McIlvain, Reading, Pa.
Henry Maddock, Bethlehem, Pa.
Charles A. Matcham, Allentown, Pa.
Mrs. Chas. A. Matcham, Allentown, Pa.
Miss C. Matcham, Allentown, Pa.
Sidney H. Matcham, Allentown, Pa.
C. A. Meissner, New York, N. Y.
Mansfield Merriman, So. Bethlehem, Pa.
Harlan C. Miner, Gloucester, N. J.
N. B. Mitman, Philadelphia, Pa.
H. S. Munroe, New York, N. Y.
R. V. Norris, Wilkes-Barre, Pa.
C. L. O'Neil, Allentown, Pa.
George Ormrod, Allentown, Pa.
Mrs. George Ormrod, Allentown, Pa.
Miss Ormrod, Allentown, Pa.
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Mrs. I. P. Pardee, Hazleton, Pa.
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G. S. Patterson, Vivian, W. Va.
Mrs. G. S. Patterson, Vivian, W. Va.
Edmund C. Pechin, Buchanan, Va.
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Mrs. J. M. Sherrerd, Easton, Pa.
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Miss Smith, Bridgeton, N. J.
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Maunsel White, Bethlehem, Pa.
W. B. Whitman, Philadelphia, Pa.
E. P. Wilbur, South Bethlehem, Pa.
Eldridge Wilbur, South Bethlehem, Pa.
W. A. Wilbur, South Bethlehem, Pa.
John F. Wilcox, Orange, N. J.
W. S. Wintersteen, Bethlehem, Pa.
Charles Ziegenfuss, Bethlehem, Pa.

Proceedings of the Ninety-First Meeting, London, England, July, 1906.*

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* Held jointly with the Iron and Steel Institute. The present report has been necessarily condensed, especially as regards excursions, entertainments and addresses, from the more extended account in the *Bi-Monthly Bulletin* of the Institute for November, 1906, pp. 809 to 908.

sop, William, & Sons, Ltd., Sheffield; Lanarkshire Steel Co., Ltd., Motherwell; Lilleshall Co., Ltd., Shifnal; Linthorpe-Dinsdale Smelting Co., Ltd., Middlesbrough; Lloyd, F. H., & Co., Ltd., Wednesbury; Macfadyen, P., & Co., London; MacLellan, P. & W., Ltd., Glasgow; Mason, Adam, & Sons, Bolton; Mining Institute of Scotland, Hamilton; Moss Bay Hematite Iron & Steel Co., Ltd., Workington; Müller, W. H., & Co., Rotterdam; Naylor, Benzon & Co., Ltd., London; Normanby Iron Works Co., Ltd., Middlesbrough; North-Eastern Railway Company; North-Eastern Steel Co., Ltd., Middlesbrough; North of England Institute of Mining Engineers; Oakes, James, & Co., Alfreton, Otis Steel Co., Ltd., London; Park Gate Iron & Steel Co., Ltd., Rotherham; Pearson & Knowles Coal & Iron Co., Ltd., Warrington; Pease & Partners, Ltd., Darlington; Pickerings, Ltd., Stockton-on-Tees; Raine & Co., Ltd., Newcastle-upon-Tyne; Richardson, Duck & Co., Stockton-on-Tees; Richardsons, Westgarth & Co., Ltd., Middlesbrough; Richmond Iron & Steel Co., Stockton-on-Tees; Ridley, T. D., & Sons, Middlesbrough; Ritchie, J. & R., Ltd., Middlesbrough; Samuelson, Sir B., & Co., Ltd., Middlesbrough; Sankey, Joseph, & Son, Ltd., Bilston; Scott, Walter, Ltd., Leeds; Seaton Carew Iron Co., Ltd., Seaton Carew; Seebohm & Dieckstahl, Ltd., Sheffield; Senior, George, & Sons, Ltd., Sheffield; Shaw, W., & Co., Middlesbrough; Simon-Carves Bye-Product Coke Oven Construction & Working Co., Ltd., Manchester; Simpson, James, & Co., Ltd., London; Skinningrove Iron Co., Ltd., Skinningrove, Carlin How, S. O.; Slag Reduction Co., Ltd., London; Smith, Frederick, & Co., Ltd., Halifax and Manchester; Somers, Walter, & Co., Ltd., Halesowen; South Durham Steel & Iron Co., Ltd., Stockton-on-Tees; Spencer, John, Ltd., Coatbridge; Steel Company of Scotland, Ltd., Glasgow; Steel, Peech & Tozer, Ltd., Sheffield; Stein, John G., & Co., Ltd., Bonnybridge; Summers, John, & Sons, Ltd., Stalybridge; Swan Brothers, Middlesbrough; Talbot Continuous Steel Process, Ltd., Middlesbrough; Teesside Bridge & Engineering Works, Ltd., Middlesbrough; Thomas, R., & Co., Ltd., Llanelly; Thornycroft, John I., & Co., Ltd., Chiswick; United States Steel Products Export Co., London; Vickers, Sons & Maxim, Ltd., Sheffield; Ward, Thos. W., Ltd., Sheffield; Warren, Beattie & Co., Ltd., Middlesbrough; Watson, James, & Co, Middlesbrough; Watson, W. J., & Co., Middlesbrough; Waverley Iron & Steel Co., Ltd., Coatbridge; West of Scotland Iron & Steel Institute, Glasgow; Whitehead, L. D., & Co., Tredegar; Whitwell, W., & Co., Ltd., Stockton-on-Tees; Wigan Coal & Iron Co., Ltd., Wigan; Williamson, R., & Son, Workington; Wilsons, Pease & Co., Ltd., Middlesbrough; Wood, John, & Sons, Ltd., Wigan; Worthington Pump Co., Ltd., London.

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Barman, H. D. D., Glasgow; Barrowman, James; Bauerman, Professor H.; Beardmore, William; Beckett, J. H., Ebbw Vale; Beesley, W. T., Sheffield; Benson, T. W., Newcastle; Bleckly, W. H.; Braby, Cyrus, London; Bright, W., Gowerton; Broadway, B. J., Birmingham; Brooks, Samuel H., Levenshulme; Brophy, Michael M., London; Brough, Bennett H.; Brown, M. Walton; Brown, P. B., London; Burnup, J. Morison, London; Burton, W., Wigan; Burtt, W. H., Middlesbrough; Bush, George, London; Butlin, W. H., Wellingborough; Cackett, J. T., Newcastle; Calderwood, James, Middlesbrough; Campbell, James, Middlesbrough; Campbell, Thomas, Glasgow; Campion, A.; Carlisle, C. G., Sheffield; Carnegie, Andrew; Carr, W. Cochrane, Newcastle; Cawley, George, London; Chalas, E. C., London; Clark, W., Glasgow; Cleeves, Frederick, London; Commans, R. E., London; Cook, Joseph, Alfreton; Cooper, Arthur; Cooper, Joseph, Jarrow-on-Tyne; Cowan, John, Cambuslang; Cowan, John; Crawshay, W. T., Reading; Cunningham, P. N., Mossend; Custodis, J. C., London; Darby, J. H., Brymbo; Davenport, F., Eccles; Davies, H., Neath; Davies, J. P., Dowlais; Davis, E. P., Ilkeston; Dewrance, John, Chislehurst; Dixon, Walter, Glasgow; Downing, N., Stockton-on-Tees; Dronsfield, W., Oldham; Dunell, G. R., Chiswick; Ellis, John D.; Etherington, John, London; Evans, John, Dowlais; Evans, William; Finlayson, Finlay, Airdrie; Firth, Ambrose, Sheffield; Firth, William, Leeds; Flint, Henry, Wigan; Forgie, J. T., Glasgow; Forster, T. E., Newcastle; Fossick, W. G., London; Foster, W. J., Darlaston; Fox, E. Marshall, London; Gaunt, J. T., Eaglescliffe; Gaunt, R., Stockton-on-Tees; Gledhill, J. M.; Gowland, Professor W., London; Gray, G. Watson, Liverpool; Greiner, A., Seraing; Gurney, R. F., London; Harbord, F. W., London; Hartlieb, J. F., London; Hatton, George, Brierley Hill; Hay, Walter R., London; Hayne, W. Crosier, London; Head, Benjamin W., London; Henderson, James, Frodingham; Hewlett, Alfred, Wigan; Hill, W., Newcastle, Staffs.; Hopkinson, R. A., Huddersfield; Horsfield, Arthur, Horbury; Horsfield, S., Consett; Howell, S. Earnshaw, Sheffield; Hoyle, J. Rossiter, Sheffield; Hughes, Colonel Herbert, C.M.G., Sheffield; Hunsiker, Colonel Millard, London; Hunter, J., Queen's Ferry; Hunter, W., Stirling; Huntington, Professor A. K., London; Jack, A.G.M., Sheffield; Jeans, J. Stephen, London; Jenkins, J. G., Glasgow; Jenkins, Thomas, Perry Barr; Jones, C. L., London; Jordan, A. J., Sheffield; Justice, P. M., London; Keen, Arthur; Keen, W. B., London; Kirkaldy, W. G., London; Lamberton, Andrew; Leigh, A. Lennox, Frodingham; Lewis, D., Dowlais; Lewis, David, Gorseinon; Levy, Lewis, London; Livesey, James, London; Longbotham, J., Sheffield; Louis, Professor Henry, Newcastle-upon-Tyne; McBean, Alex., Wolverhampton; McGregor, W., Airdrie; Mackenzie, T. B., Motherwell; MacLaren, C. F., Wishaw; McLaren, Walter S. B., London; McNeil, Charles, Glasgow; Malmberg, Olof A., Harrow; Marley, J. E., Hebburn-on-Tyne; Martin, E. P.; Massey, W. H., Twyford; Maw, W. H., London; May, Thomas, Newcastle; Mechan, Arthur, Glasgow; Mechan, Henry, Glasgow; Melling, J. F., Sheffield and Leeds; Melling, S., Wigan; Melling, Thomas, Wigan; Mellowes, Frank W., Sheffield; Meyjes, A. C., London; Middleton, J. T., London; Mills, Frederick, Ebbw Vale; Mitchell, G. A., Glasgow, Mitford, William, Redcar; Monks, F., Warrington; Moore, Alfred, London; Moore, R. T., Glasgow; Morgan, G. Vaughan, London; Morgan, P. Vaughan, London; Morgan, S. Vaughan, London; Mount-Haes, A., London; Muirhead, W., Glasgow; Munro, R. D., Glasgow; Murray, Matthew, London; Nash, Thomas, Sheffield; Needham, C. T., Manchester; Nesbit, D. M., London; Nicholson, H., Manchester; Norris, W. G., Salop; Page, Davidge, London; Palmer, Henry, Newcastle; Parrington, M. W., Newcastle; Pendred, L., London;

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Newcastle-upon-Tyne Reception Committee.

The Right Honorable The Lord Mayor of Newcastle-upon-Tyne (Alderman Sir J. Baxter Ellis), The Sheriff of Newcastle-upon-Tyne (Councillor Johnstone Wallace), President of the North of England Institute of Mining and Mechanical Engineers (T. W. Benson), Charles A. Harrison, C. S. Swan, John Tweedy, Ivan C. Barling, T. E. Foster, M. W. Parrington, J. J. Prest, F. R. Simpson, Right Honorable Lord Armstrong, J. P. Gibson.

Glasgow Reception Committee.

(The West of Scotland Mining Institute and the Mining Institute of Scotland), A. Lamberton (*Chairman*), Andrew Campion (*Secretary*), Robert T. Moore, J. G. Jenkins, W. Clark, T. B. Rogerson, Walter Dixon, G. A. Mitchell, J. T. Forgie, H. D. D. Barman, R. D. Munro.

Edinburgh Reception Committee.

John Cowan (*Chairman*).

Stewards of the Provincial Tour.

Professor H. Bauerman, W. F. Cheesewright (and Mrs. Cheesewright), D. A. Louis, A. C. Meyjes, L. Pendred (and Mrs. Pendred), Hubert S. Thomas (and Mrs. Thomas).

This joint meeting originated in a cordial invitation extended by the Council of the Iron and Steel Institute, and accepted with enthusiasm by the Council of the American Institute of Mining Engineers. Earnest invitations were afterwards received from other societies; and the Institution of Civil Engineers, the Institution of Mining and Metallurgy, the North of England Institute of Mining and Mechanical Engineers, the West of Scotland Iron and Steel Institute and the Mining Institute of Scotland co-operated heartily in the programme of the Iron and Steel Institute, as the following pages will show. Moreover, the Institute of Mechanical Engineers entertained a considerable number of guests at its meeting held July 31–August 3, at Cardiff, Wales. And finally, the Society of German Ironmasters organized and conducted a supplementary visit to Germany.

Under a mutual agreement between the Councils of the American Institute of Mining Engineers and the Iron and Steel Institute, it was decided to hold three sessions at this meeting: the first to be a session of the Iron and Steel Institute, the second, a session of the American Institute of Mining Engineers, and the third, a joint session of both Institutes.

This agreement provided further that each of the two societies should be free to publish such papers and discussions, presented at either of the sessions, as it should deem desirable. In accordance with this understanding, certain papers presented to the Iron and Steel Institute have been published in the *Bi-Monthly Bulletin* of this Institute; and some of the most important are also included in this volume. For the text of papers not thus reprinted by this Institute, members are referred to the *Journal of the Iron and Steel Institute.*

The first session, being a session of the Iron and Steel Institute, was held in the Hall of the Institution of Civil Engineers, 12 Great George Street, Tuesday, July 24, at 10 a.m., President Hadfield presiding.

Addresses of welcome were delivered by President Hadfield and by Sir James Kitson,* senior past-President of the Iron and Steel Institute, to which President Hunt, of the American Institute of Mining Engineers, made an appropriate reply.†

* *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 816–824.

† [illegible] 824–825.

A Diploma of Honorary Membership in the Iron and Steel Institute was presented by President Hadfield to Professor Ehrenwerth, of Leoben, Austria.

The second session, being a session of the American Institute of Mining Engineers, was held at the same place, Wednesday, July 25, at 10.30 a.m., President Hunt presiding. Mr. Hunt delivered the following address:

President Hunt's Address.

While the actual dates of the organization of the American Institute of Mining Engineers and the introduction into America of the Bessemer process were not exactly identical, at the same time they were so close together that their histories have run largely parallel. Therefore it seems to me to be fitting upon the occasion of our holding this meeting in England, the birthplace of the Bessemer process, to give some account of its progress in America.

Unless inspiration is vouchsafed, prophecy is a very dangerous venture, and probably in no sphere has that danger been more clearly defined than in relation to the iron and steel industry. So frequently has it seemed as though the summit of all possible achievements had been reached, that the most astute minds have ventured to assert this conclusion. Again, physical conditions have apparently been so clearly unalterable that deductions as to the possible and impossible have seemed safe. This has been notably the case in America. It will be recalled that in the 'seventies a distinguished iron metallurgist, and a gentleman whom it has been the pride of both hemispheres to honor, visited America and carefully studied the iron and steel situation, with the result that he unhesitatingly proclaimed that the development of her possible production of iron, and her influence in the markets of the world, were plainly limited by geographical conditions to such an extent that the old world need not fear her rivalry. When this prophecy was made it seemed to be absolutely logical, and based upon conditions which could not be altered or overcome. Indeed, he gave actual figures showing that the transportation distances were so great over which it would be necessary to bring iron-ore and the fuel to smelt it to a common point, and then after

its reduction the transportation of the products to market would again cover such distances, that it was impossible for successful commercial competition to be created. At that time the statement that iron-ore could be transported over one hundred miles of railway and eight hundred of water and placed at the then and now very center of the American iron industry at a cost of $2.40 per ton, would have been received as the wildest lunacy. That has all been accomplished, and instead of having brought disastrous results to the transportation interests, it has, on the contrary, yielded such profits that they have been built up to colossal proportions. And the finished product can be placed at the seaboard for foreign markets at transportation-cost little, if any, greater than is required for internal transportation in many European countries.

These low carrying-charges have been co-relative with a tremendous ore development in the Lake Superior region, which has steadily increased until in 1905 there were taken from there 34,353,456 gross tons. While there still remain many proved millions of tons of ore, it is recognized that such a production cannot be indefinitely maintained. This condition is leading to increased interest being taken in the other ore sections, some of which, while well known, have been unworked, either because of location or of comparatively low iron-percentage of the ore. The mineral resources of the Southern States impressed Sir Lowthian Bell, when he was investigating iron-ore conditions, and he predicted a prosperous future in that region. For some years it seemed as though such hopes were doomed to disappointment; but there is no longer any doubt of the wisdom of the prediction. In 1880 the pig-iron production of all the Southern States was 387,000 gross tons. In 1905 it was about 3,100,000 gross tons. At the same time, it has not been in iron alone that the South has grown during the past twenty-five years. In 1880 there was invested in cotton-mills about $21,000,000. In 1905 it had increased to $225,000,000. During the same period the capital invested in the Southern States in all kinds of manufactures increased from $257,000,000 to $1,500,000,000.

As might be expected, with this great revolution in cost of transportation there has also come a revolution in selling-prices.

While the political policy of the United States has tended to maintain a higher range of prices than would probably have prevailed under other conditions, at the same time it is true that, during a time of severe business depression, steel rails were made from ore mined at the head of Lake Superior, smelted with coke from Pennsylvania coal, transported about 500 miles to Chicago, and there sold, delivered, at $15 per gross ton, and at that price, under the then-existing conditions, were within the cost of manufacture.

It is most earnestly to be hoped that such times may not return, but if, under pressure, that could be accomplished some years ago, there is no reason to suppose that under similar stress the history could not be repeated. I mention this incident merely as a matter of record.

It will be recalled that the first heat of Bessemer steel made in America was produced in an experimental plant in Wyandotte, Mich., in the autumn of 1864, and that the first commercial rolling of steel rails was in the mills of the Cambria Iron Co., at Johnstown, Pa., in August, 1867, from steel made by the Pennsylvania Steel Co., at Steelton, Pa. The development of the business was very rapid, but not always attended with profitable results.

In June, 1876, there were ten rail-mills in operation, and an eleventh nearly ready to start. At that time one Bessemer company had already gone to the wall. One of these ten companies, as well as the eleventh, and their works, have absolutely gone out of existence. In fact, but five of them are now making rails. One of these has but a limited production in that department, and the other four are making their rails in mills not then in existence, and in one case in a mill hundreds of miles away. It is also true that three of the then-separate corporations have since been consolidated under a new title into one, which, with its new mill, is enumerated in the existing five; and this mill is situated some miles from the location of either of the original ones. Two of the other companies, while actively at work in other lines, have ceased making rails.

Since 1876, in addition to those already mentioned, seventeen corporations have erected mills to roll standard-weight steel rails, thirteen from steel of their own manufacture, and four from purchased blooms. Seven of the steel producers are

now making rails, two are on other products, and the remaining eight have gone out of existence, so that there are now in the United States ten corporations running thirteen rail-mills. Of these, one company is rolling its rails from basic open-hearth steel; and another one from either that or Bessemer. I only refer to mills rolling rails of 60 lb. and over per yard.

Three of the companies are controlled by one parent corporation; three others by another, and two others by still another; thus leaving two single and independent concerns. In addition, the building of another mill of large capacity, to roll basic open-hearth rails, with the required blast-furnaces, steel furnaces, and town, is actively under way; this being done by one of the subsidiary companies of the United States Steel Corporation. The contemplated expenditure will amount to about $50,000,000.

In 1876 Canada did not possess a steel-rail mill of any kind. To-day there are two, situated about 1,500 miles apart, one practically on the shores of the Atlantic, and the other at the foot of Lake Superior. They naturally depend upon distinctly different sources for their supply of ore, fuel, etc. The Eastern works are producing basic open-hearth rails from iron made in their own blast-furnaces from near-by ores and fuel. The Western one is so far depending upon the Bessemer process, but has one basic open-hearth furnace about completed. As yet, the ores used by it are from Lake Superior deposits located in the United States, and the coal and coke are brought from the States of Ohio, Pennsylvania, and Virginia. That this is at all possible, well illustrates how low transportation-charges have been brought. One of the two blast-furnaces of this establishment is run on American ores and Canadian charcoal, and is making a record production. Both works are operating successfully, and enjoying bounties from the Canadian Government, but as yet their production is not equal to the immediate demands of the Dominion.

While few words have been required to tell of the various changes named above, the attendant incidents have been startling and the results tremendous. In ten cases absolutely new communities have been organized, and these have grown to embrace thousands of people.

While I have not the actual tonnage of the rail-production

of the several works in the earlier days, I have some of the records of their ingot-output, and as at that time rails were practically their only finished product, a comparative idea can be formed. Moreover, we do know the total rail-production of the country for the various years.

In a paper presented to the American Institute of Mining Engineers in 1876,[1] in which my patriotism sought vent in words, I pointed out some of the salient improvements in the mechanical details of American Bessemer plants which had made possible the then great increase in their production, and rightfully credited most of them to Alexander L. Holley, while at the same time giving what seemed to me due credit to others who up to that time had been prominent in the business.

It is a well-known matter of history that Mr. Holley was the first one to bring to America the Bessemer process in what I may call an organized condition, and that after several years of legal skirmishing a consolidation of the patent interests involved was effected, following which the erection of works to make steel by that process was quite rapid, and, with the exception of three, Holley acted as consulting engineer during their construction, and in fact during their subsequent operation, this connection lasting until his death in 1882. Two of the works with which he was not connected made disastrous financial failures, and while the other one was successful, it never ranked with several of its rivals in the matter of volume of output. So it was that Alexander L. Holley's personality was indelibly stamped upon American Bessemer-steel making. Fortunately, his efforts were joined by those of a number of exceptionally capable men. The names of Holley, John Fritz, George Fritz, William R. Jones, Daniel L. Jones, and Robert Forsyth furnish a galaxy of talent which makes plain why America so soon forged to the front in Bessemer-steel production.

It will be recalled by the older of my foreign hearers that in his capacity of consulting engineer to the American Bessemer manufacturers, Holley made at least yearly visits to Europe, where, largely by reason of his charming and lovable personality, he received free access to practically all steel-works, and

[1] *Trans.*, v., 201 to 216 (1876–77).

it was well understood by his hosts that he was seeking information to be used by his American clients. It was also known that, on the other hand, he was ready to give freely in return the best he had. Holley was peculiarly fitted for this post. The information he gained by observation was not used in mere copying, but rather served as a basis on which to build. The freedom of his mind from prejudice was well shown by his position toward the open-hearth processes, both acid and basic. Holley, in spite of his belief in and devotion to the Bessemer process, strongly urged upon his American clients the claims of the others; and it is historical that on the occasion of one of the Institute's meetings, when jokingly reminded by some of his Bessemer associates that it seemed strange for such advocacy to emanate from him, he replied: "You may think me crazy, but I believe that some of you will live to see the open-hearth process attend the funeral of the American Bessemer." The death has not yet occurred, and alas! he and many of his listeners have passed away, but some still live and know that some members of the American Bessemer family have at least begun to realize that the limitation of production by that process has been reached.

In my paper before mentioned I called attention to the invention by Mr. George Fritz, then chief engineer of the Cambria Iron Co., Johnstown, Pa., of the blooming-mill on which to roll steel ingots to blooms instead of reducing them by hammering, as had been the practice. This was first accomplished in 1867 on a modified rail-train of rolls. Mr. Fritz built his first regular three-high blooming-mill in 1871. This departure from the old practice, added to Holley's modified converting-plants, greatly helped to increase production.

At the time of my Philadelphia paper, June, 1876, the Bessemer plant of the North Chicago Rolling-Mill Co., built from Holley's plans, and then under the charge of Mr. Robert Forsyth, held the record for a month's production of ingots at 6,457 gross tons. During the year 1876, there were made in the whole United States 469,639 gross tons of Bessemer ingots, from which 368,299 gross tons of rails were made, selling at an average price of $59.25 per ton with gold at $1.10.

There was a constant increase in production until 1887, when that of ingots reached 2,936,035 tons and rails 2,101,904 tons;

the latter sold at an average price of $37.08 per ton, with currency at par. The year 1887 was one of unparalleled railroad-building in the United States, followed by a period of reaction, with the result that while the output of ingots kept up, that of rails fell off, and it was not until 1899 that the rail-production again passed the two-million-ton point. In that year there were turned out 7,586,354 tons of Bessemer ingots, 2,947,316 tons of open-hearth ingots, and 2,270,585 tons of rails, all of Bessemer steel, the latter selling for an average price of $28.12 per ton. Following that year there was a continued increase in the output of open-hearth steel, while that of Bessemer remained more nearly constant; until in 1905 there were made 10,919,272 gross tons of Bessemer ingots, 8,444,836 gross tons of open-hearth ingots, and 3,375,611 gross tons of rails, 183,264 tons of which were of basic open-hearth steel. The rails were practically all sold at a uniform price of $28 per ton, which has been the standard price since, and including, 1902.

I have referred to the changes which have taken place in the organizations of steel companies and the location of their plants, and I have stated that the North Chicago Rolling-Mill Co. held the monthly record for product of Bessemer ingots in 1876. In 1882 this company built an entirely new Bessemer and rail plant at South Chicago, about fifteen miles away from its old one. The converting-works were designed and erected by Mr. Robert Forsyth. In 1889 these works were included in the consolidation forming the Illinois Steel Co. Later the North Chicago plant was dismantled. At first the new rail-mill was a reversing one, but following its acquisition by the new company Mr. Forsyth, who, after an interim of several years, then returned to the management, entirely remodeled it by putting in a three-high mill with automatic tables. There have been changes in the management and additions to the plant since then, but, fundamentally, the converting-works and mill are the same, and their record production is 91,424 gross tons of ingots and 71,424 gross tons of rails in a month. All of these rails were rolled on one rail-mill. It consists of three sets of rolls, beside the blooming-train, set in échelon, but making one mill, all the steel being reduced through the same passes in the rolls.

In 1886 the rail-mill of the Edgar Thomson Works had been

doing great work, but it was being pressed in output by other mills, and Mr. Andrew Carnegie then, as on many other occasions, displayed his acute business acumen, and directed Captain Wm. R. Jones, manager of the works, to prepare plans for the very best rail-mill he knew how to design. Captain Jones, in one of our many intimate talks, told me that some time afterwards Mr. Carnegie asked him how he was progressing with his plans, and, on his reporting, asked how much the mill would cost. The Captain replied that he could not then tell. Mr. Carnegie said, "Well, but we must place some limit on its cost!" Jones answered, "You told me to design the very best mill in my power; now if I am to be limited by the cost-sheet in so doing, I must give up the job." Mr. Carnegie then asked, "But if we build such a mill, how much will you promise to increase its production over the present one, and how much per ton will you save?" Jones answered, "I will promise to double the output, and save 50 cents per ton." "All right," was the answer, "go ahead, and do your best." The new mill started in 1888, and all promises for it have been much more than fulfilled. Its record production of rails for a month is 61,033 gross tons. There are three sets of rolls in this mill, which are placed tandem. It has been strengthened since Captain Jones's death, but is practically his mill. The converting-works, which were originally designed by A. L. Holley, afterwards altered and added to by Jones, and again by C. M. Schwab, have made about 105,000 gross tons of ingots in a month. After the starting of the new mill, the original Edgar Thomson rail-mill remained idle for several years. It was then remodeled and used for the production of rails, mostly under 60 lb. per yard, and has been in practically constant operation ever since. Within the last year another rail-mill has been added to the plant, and one which is quite a departure in the business. It consists of two sets of 18-in. rolls, placed tandem, and equipped with automatic tables, the power for both rolls and tables being electric. It is intended to use this mill for the reduction of second-quality rails of standard sections, made in the other mills, to lighter ones, but as yet it has been principally used for rolling small-sectioned rails from billets.

Another notable happening in American rail-making history was the consolidation of the Lackawanna Iron & Steel Co.

and the Scranton Steel Co., both of which were located in Scranton, Pa. Following it, the rail-mill of the former company was abandoned, all the work being given to the latter, which was of more recent construction, and was of the reversing type, and with one other constituted the only existing examples of such mills in America. The other one was built by Mr. A. J. Moxham, as President of the Johnson Co., in 1888, near Johnstown, Pa., to roll steel-girder street-car rails. In 1894 this mill was removed to Lorain, Ohio, and is now operated by one of the subsidiary companies of the United States Steel Corporation. While its principal output is girder and other street-car rails, a large tonnage of standard shapes is also produced.

The Lackawanna Steel Co. decided to abandon Scranton as a manufacturing point, selecting a new location at West Seneca, N. Y., on Lake Erie, near the city of Buffalo. In placing the rail-mill in its new location some changes were made, but the reversing type was retained. Rail-rolling was resumed upon it in October, 1903. It is somewhat remarkable that the reversing rail-mills in America should have developed such a migratory disposition. I am certain from personal knowledge that in neither case was it caused by unsatisfactory foundations. It is a matter of record that when putting in the foundations at Scranton, Pa., Mr. W. W. Scranton, who built the works, made an innovation on American engineering practice by using concrete, instead of stone or brick, in its foundations, and with such complete success that the practice soon became general.

I have mentioned the record tonnage of but two plants, because they have been the largest, and strikingly illustrate the great increase in American output. But all the other American works are running splendidly.

Among the mechanical improvements of the Bessemer converting plants, which did much to increase their output, was the abandonment of the casting-pit, and the adoption of the present practice of filling the molds while standing on cars, which are immediately thereafter pulled outside of the converting-works, and subsequently, when the steel has sufficiently cooled, the molds are mechanically stripped from it, thus saving much manual labor and time; in fact, without this improvement in practice the present output would be impossible. Holley's shallow pit was the first step—this, seemingly, the final

one. I do not know which works first adopted the plan, but it will be found by reference to the Institute's *Transactions*, vol. xiii., that at its New York meeting, February, 1885, L. G. Laureau read a paper in which he proposed so casting and handling the steel. He called attention to the fact that both in England and on the Continent ingots were sometimes cast from a ladle into molds placed on cars, and said, "I believe the solution of the problem lies in casting the ingots into molds placed on cars so constructed that all subsequent operations, such as stripping and putting back into place, may be done automatically, or by easily-handled machinery." Further: "The pit and the ingot-cranes can be entirely suppressed, and all the operations of casting, cleaning, ladle-changing, etc., can take place on the general level." But his plan was to turn the filled molds on their sides, and force the ingots out by a hydraulic plunger. Captain Jones subsequently adopted at the Edgar Thomson Works that part of the proposed practice. But it was found that with the larger-sectioned ingots, which had come into use, this hasty placing of them on their sides increased the prevalence of pipes; so it, as well as the use of horizontal heating-furnaces, was abandoned.

Another factor in increasing output was the use of metal direct from the blast-furnaces. This not only added to production, but at the same time decreased cost by saving the expense of re-melting; but it was not entirely successful until the mixer invented by Wm. R. Jones was adopted. The claim that the credit of this, as an invention, belonged to him was after his death bitterly fought, and the case carried to the U. S. Supreme Court, which decided in favor of his claim; therefore it must be so considered. That he was the first to venture to accumulate 150 tons and over of molten metal in a refractory-lined vessel, to be drawn therefrom as wanted and taken to the converters, has never been denied.

The Edgar Thomson works, then under his management, were running on direct metal, and experiencing the usual troubles incident to its use. In seeking for a way to eliminate them, he planned the mixer. This was soon after the works began the use of natural gas, and he expected to be compelled to rely upon heat from it to keep his iron sufficiently hot. He also thought it might be necessary to agitate the metal while

in the mixer to insure sufficiently uniform results; and so designed his apparatus. Neither procedure was found necessary. I have wondered if the use of natural gas had not been possible, would he have made the venture? Undoubtedly the invention would have come in time, but probably would have been much delayed.

It is with some modest hesitation that I refer to the part which mechanical appliances at the rolls have performed in the great increase in steel-rail output; but without them it would have been impossible. Until March, 1884, all American rail-mills were fed by the use of hooks and tongs; and three-high trains required from fifteen to seventeen men to operate them for a production limited to 300 tons per turn of twelve hours. Numerous inventors had sought to accomplish this work by machinery which would be automatic in its action, but I believe until that time there had not been any actually built. Its possibility was discussed and predicted; and indeed, as I have recorded on another occasion, Holley said, in that spirit of prophetic jest so constant and so charming in him, "that the day would come when we would start a rail-mill on Monday morning, and then go home after locking the doors, only returning each morning to count the rails that had been made during the preceding twenty-four hours, no other manual labor being necessary." That point has now been reached, and in fact the output has become so rapid, that if the count were not kept as the rails are made, I fear the enumerator would never catch up. How Holley would have reveled in the knowledge of what is being accomplished! He knew that rail-mill mechanism was possible, but for some incomprehensible reason the way did not open to his mind.

In March, 1884, I introduced driven tables in front of the finishing-rolls of the rail-train of the Albany & Rensselaer Iron & Steel Co., Troy, N. Y. They worked so well that I put an automatic arrangement in front of the roughing-rolls. This was more particularly designed by Mr. Max M. Suppes, then the master mechanic of the department, and now general manager of the Lorain works of the U. S. Steel Co. Mr. Suppes rendered me valuable assistance during all my experiments. The last table was also successful, and soon after I placed tables on the catchers' side of the train.

Capt. Wm. R. Jones at once advised his firm to secure authority and apply the system to the Edgar Thomson mill. This being done, he designed and put in an elaborate system of tables, some points of which he patented. He, Mr. Suppes, and myself pooled our interests, and later nearly all of the steel companies of the country secured licenses from us. The number of men necessary to operate the rolls was at once reduced from seventeen to five, and I have already given the figures covering the increased production. From 300 to 3,000 tons per day is a far cry.

Others took up the automatic-table matter, and Mr. F. H. Treat, then mechanical engineer of the Joliet Steel Co., put in a set after his designs at their Joliet, Ill., mill. He was assisted in this by Mr. Charles Pettigrew, the company's chief engineer. Mr. Wm. Clark, of Pittsburg, also developed a table system. As before stated, there have been great changes in the various companies' mills, but the table designs are all based on the original schemes.

I have mentioned by name but few persons, and the majority of those are either deceased or not now actively connected with steelmaking. But this does not mean that all credit belongs only to those so named. Far from it. There has been and is now an army of workers, some of whom have reaped rich pecuniary rewards; others are still in the hottest of the conflict, no doubt hoping for personal success and emoluments, but, over and above all other considerations, giving their best toward the success of their works. I cannot mention them all, and whom could I select?

It must be remembered that these men have improved and strengthened the plants which they manage or have designed, and with that and their administrative efforts have come the great results. To be content with conditions as they found them would have been to such men impossible.

Demand creates supply; therefore there would not have been the increased steel production in America if it had not been for the development and growth of the country, and it is collaterally true that without the discovery and expansion of the making of Bessemer and other mild steels that growth would have been impossible. While the present American railways with their equipment would be absolutely impossible with iron rails,

it is also true that their development has necessitated a change in the characteristics of the steel rails used. In the early 'seventies, when steel rails began to replace iron ones, the section required was not over 60 lb., and more generally 56 lb. to 58 lb., to the yard, and the increased service obtained from them was regarded as wonderful. But, nevertheless, the wisdom of their use was not at once unanimously accepted. If I am not mistaken, one of the leading English technical journals for a long time warmly championed the continued use of iron rails. Naturally, the term "steel" carried with it an idea of hardness, and consequent brittleness; hence the fear of an all-steel rail. Much time and money were spent in unsuccessful experiments with iron ones having steel heads. These conditions led to making the steel rails of as soft a composition as could then be successfully accomplished. I have no doubt that had ferromanganese been then known the early rails would have been rolled from dead soft metal. Let this be as it may, it is true that while the early steel rails were of very irregular chemical composition, the aim was to keep the carbon-content not to exceed 0.30 per cent., and somewhat later not over 0.40 per cent. Undoubtedly, owing largely to the care exercised in the physical treatment of the metal during the manufacture of the rails. most excellent results were obtained, and gradually familiarity with them dispelled the fear of breakage. And it was not long before railway officials realized that their use permitted increased loads and speed. Traffic demands and better financial conditions led to increasing the weight of the rail-sections, and also the hardness of the metal in them; but the same causes also led to heavier wheel-loads of both engines and cars, and faster schedules. In many cases the results as to wear were not satisfactory, which led to a notable discussion, which is recorded in the *Transactions* of the Institute. The advocacy of softer steel came from one of the leading railway systems of the country, ably presented by their chief chemist, then, as now, a distinguished member of the Institute. This resulted in a demand for lower-carbon steel, which continued for several years. But the results were not as anticipated. The increase of wheel-loads, total tonnage hauled, and speed of trains went on, and harder steel won the day, and has since been in universal use on American roads. But the enlargement of elements

was not all on the side of the railways. The railmakers had also been busy, and some persons claimed, and still contend, that speed of manufacture did not tend to give better results under stress of traffic. Let that be as it may, it is a fact that American railways now require very much harder rails than do those of Europe, and this regardless of the climatic conditions under which the rails are to be used. In fact, thousands of tons of 80-lb. rails, with carbon-content to the height of American specifications, have been, and are, in safe use on Canadian roads, some of them made on this side of the water, some in the United States, and others in Canada.

The increase in the output of basic open-hearth steel has been great in the United States; but it was not until 1899 that the commercial manufacture of rails from that metal was begun. This was done at the Ensley, Ala., works of the Tennessee Coal, Iron & Railroad Co., and since then that company has regularly continued the manufacture.

The Colorado Fuel & Iron Co., at Pueblo, Colo., has begun the manufacture of basic open-hearth rails on a large scale.

As I have already mentioned, the entire rail output of the Dominion Steel Co., of Sydney, Nova Scotia, is of that metal. And the Indiana Steel Co., a subsidiary company of the United States Steel Corporation, has started upon the building of a large plant for the same purpose. Aside from the comparative merits of Bessemer and basic open-hearth steel, there seems to be no doubt but that America's iron-ore conditions will force the growth of the latter process. As I have already stated, in 1905 there were rolled in the United States 183,264 tons of basic open-hearth rails, which is small as compared to the Bessemer tonnage, but it is "the handwriting on the wall!" There have been two attempts to make steel in America by the basic Bessemer process. Both were technically successful, but, owing to the character and cost of the iron obtainable, they failed commercially.

While so far the output has covered but a moderate tonnage, at the same time the successful results obtained demand that mention be made of the McKenna process of renewing worn rails. The American McKenna Process Co. owns three mills, but at present only one of them is in operation. It is situated at Joliet, Ill., and has been continuously successful. As you

will recall, the procedure is to take worn rails, heat them in long furnaces, and give them, while at a comparatively low temperature, two passes in a tandem mill. The reduction varies with the condition of the worn section, but, as a rule, gives a rail of about from 10 per cent. to 12 per cent. lighter weight than the original section. This practice has been in operation since 1897, and a number of the leading Western railway systems are regularly employing it with satisfactory results.

Another line of steel manufacture has had a great growth in the United States. That is the rolling of structural shapes, practically all of which are of steel. Naturally the demand, and so the output, has varied. I find that in 1902 there were rolled 1,300,326 tons; in 1904 it fell to 949,146 tons; while in 1905 it was 1,660,519 gross tons, being 28 per cent. over 1902 and 74.9 per cent. over 1904.

While I have given precedence to the American development of Bessemer and open-hearth steels, particularly as intended for rails, it was because, as stated, that history has been so coincident with that of our Institute. But it has not been on those lines alone that America has made great progress. In fact it was necessary to such production of steel that increase in the blast-furnace output should at least keep pace with it. The ore had to be smelted preparatory to the subsequent processes. It will be recalled that European and American ironmasters differed for a long time as to the economy of forcing the workings of blast-furnaces, the former contending that while the output would be augmented, so also would the cost of repairs. On the other hand, the Americans maintained that it was as to the cost per ton, and not as to time, that such cost should be figured. That is, if a furnace-lining gave, say, 100,000 tons of metal and lasted but a year in so doing, it was cheaper thus to use the plant than to take three years in obtaining the same product. At all events, it has been on the latter lines that the business has been conducted. But, as in many other cases, we had to come to England for that which has made such driving possible. Without the firebrick stoves it could not have been accomplished. They may have been improved, and other names deserve honor for what has been done, but that of Whitwell will ever stand as the foundation one.

The production of pig-iron of all kinds in the United States

in 1876 was 1,868,961 gross tons. In 1905 it was 22,992,380 gross tons. The production of Bessemer pig-iron was not separated statistically from other pig-iron until 1887. In that year it was 2,875,462 gross tons, and in 1905 it was 12,220,209 gross tons. The production of basic pig-iron was first separately ascertained in 1896, when it was 336,403 gross tons. In 1905 it was 4,105,179 gross tons, charcoal basic pig-iron not being considered in either case.

Undoubtedly the use of the Jones scheme of a mixing receptacle has done much to permit the driving of the blast-furnaces supplying steel plants. The output of several furnaces being so treated, it is readily seen that greater variation in its character can be permitted than if the iron from each one had to be used separately. But while furnaces have been driven to making an output of over 750 tons per day apiece, I believe it has been concluded that better results are obtained by limiting the output of the same furnaces to about 550 tons per 24 hr. Of course, the tremendous outputs would be impossible if the raw materials, and in fact the produced metal, were not handled by machinery, much of which is of automatic character. Such devices do not merely move the stock, but some of them also regulate the charging and distribution of it in the furnace, and contribute to the regularity of the metallurgical process. In that field there has been much mechanical ability applied, and improvements are still being diligently sought. Next to the use of gas blowing-engines, and for which we must thank this hemisphere, I suppose the development in blast-furnace practice which is attracting the most attention in America is our past-President James Gayley's application of refrigeration to the blast, the results from which are certainly very encouraging, and I am glad to know that its use is being taken up by other concerns than his own.

Gentlemen, there are many other developments in the iron and steel art in America which deserve recognition, and I feel derelict in failing to mention any one of them, but time must limit my address. American members of the American Institute of Mining Engineers know of them. Our foreign members and our hosts of the Iron and Steel Institute must come to America and see for themselves. They may not want to copy, but they can, at least, see what to avoid.

At the conclusion of his address, President Hunt announced the election, as Honorary Members of the American Institute of Mining Engineers, of Messrs. R. A. Hadfield, President of the Iron and Steel Institute, and John E. Stead, F.R.S., of Middlesbrough, both of whom acknowledged in brief remarks the honor thus conferred.

The third and concluding session, being a joint session of the American Institute of Mining Engineers and the Iron and Steel Institute, was held at the same place, Thursday, July 26, at 10.30 a.m. At the request of President Hadfield, President Hunt presided.

The following papers were presented in oral abstract by their authors:

The Roe Puddling Process, by James P. Roe, Pottstown, Pa.*

Mr. Roe's paper was orally discussed by Dr. Raymond, Axel Sahlin, E. S. Cook, F. W. Paul, Prof. H. Bauerman, Prof. Turner, and B. Talbot.

Improvements in Rolling Iron and Steel, by James E. York, New York, N. Y.

Mr. York's paper was orally discussed by President Hunt, Kurt Kerlen, and James Riley.

Comparison of American and Foreign Rail-Specifications; with a Proposed Standard Specification to Cover American Rails Rolled for Export, by Albert Ladd Colby, New York, N. Y.

Mr. Colby's paper was discussed orally by Messrs. Raymond, Windsor Richards, Williams, Harbord, Hadfield, Stead, York, and Lamberton, and in correspondence by Messrs. Kenney, Sauveur, Freir, Webster, Palmer, and Nigond.

The Gas-Producer as an Auxiliary in Iron Blast-Furnace Practice, by R. H. Lee, Liberty Furnace, Va.

Mr. Lee's paper was orally discussed by Messrs. Pullon and Havard, and, in correspondence, by Prof. Wm. Kent.

The following papers were presented in oral abstract by the Secretary in the absence of the author:

The Design of Blast-Furnace Gas-Engines in Belgium, by Professor H. Hubert, Liège, Belgium.

The Application of Large Gas-Engines in the German Iron and Steel Industries, by K. Reinhardt, Dortmund, Germany.

* Not included in this volume, but published in the *Journal of the Iron and Steel Institute*.

The following paper was presented in oral abstract by the author:

Notes on Large Gas-Engines Built in Great Britain, and Upon Gas-Cleaning, by Tom Westgarth, Middlesbrough, England.

The above papers on gas-engine practice were orally discussed by Messrs. Greiner, Westgarth, Raymond, Kent, Duff, Hamilton, Tannett-Walker, Robertson, and Thwaite.

During the sessions the following papers of the American Institute of Mining Engineers, in pamphlet form, were distributed:

Piping and Segregation in Steel Ingots, by Henry M. Howe, New York, N. Y.

Effect of Low Temperature on the Recovery of Steel from Overstrain, by E. J. McCaustland, Ithaca, N. Y.

A Simple Rotary Distributor for Blast-Furnace Charges, by David Baker, Philadelphia, Pa.

A New Colorimeter for the Determination of Carbon in Steel, by Chas. H. White, Cambridge, Mass.

Internal Stresses and Strains in Iron and Steel, by Henry D. Hibbard, Plainfield, N. J.

Heat-Treatment of Steels Containing Fifty Hundredths and Eighty Hundredths Per Cent. of Carbon, by C. E. Corson, Latrobe, Pa.

Methods of Mining, Hauling and Screening at the Mines of the Aldrich Mining Co., at Brilliant, Ala., by T. H. Aldrich, Jr., Birmingham, Ala.

The following papers of the American Institute of Mining Engineers were read by title:

The Washoe Plant of the Anaconda Copper-Mining Co. in 1905, by L. S. Austin, Houghton, Mich.

The Tin-Deposits of the Kinta Valley, by William R. Rumbold, Oruro, Bolivia, So. Am.

The Amalgamation of Gold-Ores, by Thomas T. Read, New York, N. Y.

The Lime-Roasting of Galena, by Walter R. Ingalls, New York, N. Y.

The Clays of Texas, by Heinrich Ries, Ithaca, N. Y.

A Device for Regulating the Discharge of Water from a Reservoir, by P. Bouéry, Weaverville, Cal.

The Cyanidation of Raw Pyritic Concentrates, by F. C. Smith, Wenden, Ariz.

A Study in Refining and Overpoling Electrolytic Copper, by H. O. Hofman, R. Hayden and H. B. Hallowell, Boston, Mass.*

The Constitution of Ferro-Cuprous Sulphides, by H. O. Hofman, W. S. Caypless and E. E. Harrington, Boston, Mass.*

Laboratory Experiments in Lime-Roasting a Galena-Concentrate with Reference to the Savelsberg Process, by H. O. Hofman, R. P. Reynolds and A. E. Wells, Boston, Mass.*

A Search for the Causes of Injury to Vegetation in an Urban Villa near a Large Industrial Establishment, by Persifor Frazer, Philadelphia, Pa.*

Discussion of Paper by Mr. Watson on Lead- and Zinc-Deposits of the Virginia-Tennessee Region, by Frank Firmstone, Easton, Pa.

The following papers of the Iron and Steel Institute, in pamphlet form, were distributed. Of these only the paper of Messrs. Osmond and Cartaud has been included in this volume; the others are published in the *Journal of the Iron and Steel Institute.*

The Crystallography of Iron, by F. Osmond and G. Cartaud, Paris, France.

The Influence of Silicon and Graphite on the Open-Hearth Process, by Alex. S. Thomas, Cardiff, Wales.†

The Kjellin Electric Steel-Furnace, by E. C. Ibbotson, Sheffield, England.‡

The Constitution of Iron-Carbon Alloys, by Albert Sauveur, Cambridge, Mass.§

Tempering and Cutting-Tests of High-Speed Steel, by H. C. H. Carpenter, Teddington, England.

Recent Processes in Machine-Molding Practice, by Ph. Bonvillain, Paris, France.

Different Modes of Blast-Refrigeration and Their Power Requirements, by J. E. Johnson, Jr., Longdale, Va.

The Nodulizing and Desulphurization of Fine Iron-Ores and Pyrites-Cinder, by Albert Ladd Colby, New York, N. Y.

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* Manuscript not received in time for publication in this volume.

† Also *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 931–938.

‡ Also *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 967–970.

§ Also *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 939–966.

MEMBERS AND VISITORS REGISTERED AT LONDON.

The following list, which in all probability does not contain the names of all who attended the sessions and excursions, is composed of the names of members and guests registered at headquarters:

Mr. Edwin Adams, Manchester.
Mr. E. Adamson, West Hartlepool.
Mrs. E. Adamson, West Hartlepool.
Mr. Harold Adamson, Hyde.
Mr. Joseph Adamson, Hyde.
Mrs. Joseph Adamson, Hyde.
Mr. William Affleck, Ellsworth, Pa.
Mr. E. T. Agius, London.
Mr. G. Ainsworth, Consett.
Miss Ainsworth, Consett.
Mr. Samuel Allen, Edgbaston.
Mr. William Edgar Allen, Sheffield.
Sir J. T. N. Alleyne, Bart., Belper.
Miss Alleyne, Belper.
Mr. M. H. Allott, Rotherham.
Mrs. M. H. Allott, Rotherham.
Mr. James A. Anderson, Greenock.
Mr. E. G. Appleby, London.
Mrs. E. G. Appleby, London.
Mr. Robert Armitage, Leeds.
Mr. Thomas Ashbury, Manchester.
Mrs. Sarah Ashbury, Manchester.
Mr. John A. F. Aspinall, Liverpool.
Mr. Albert J. Astbury, Birmingham.
Mrs. Albert J. Astbury, Birmingham.
Mr. L. H. Atkinson, London.
Mr. A. M. Austin, New York, N. Y.
Mrs. A. M. Austin, New York, N. Y.
Capt. R. K. Bagnall-Wild, London.
Mrs. R. K. Bagnall-Wild, London.
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Mrs. Thomas H. Bailey, Birmingham.
Mr. Hugh A. Bain, Riverside, Cal.
Mrs. Hugh A. Bain, Riverside, Cal.
Mr. H. Kelway Bamber, London.
Mr. Ernest Jefferson Barnes, Sheffield.
Mr. Geo. D. Barron, Rye, New York.
Mrs. Geo. D. Barron, Rye, New York.
Miss Dorothy Barron, Rye, New York.
Miss M. Elena Barron, Rye, New York.
Mr. Herbert Bates, Manchester.
Mr. William Ralph Bates, Derbyshire.
Mrs. William Ralph Bates, Derbyshire.
Prof. H. Bauerman, London.
Mr. Jacob Becker, Kalk, near Cologne.
Mr. J. C. Beeley, Hyde, near Manchester.
Dr. Robert Bell, Ottawa, Can.
Mrs. Robert Bell, Ottawa, Can.
Sir Hugh Bell, Bart., London.
Lady Bell, London.
Miss Benson, London.
Mr. R. S. Benson, Thornaby-on-Tees.
Mr. G. H. Bentley, Manchester.
Miss L. Bentley, Manchester.
Mr. E. Bernheim, Pittsburg, Pa.
Mr. Clarence Bird, London.
Mrs. Clarence Bird, London.
Mr. Fred. Bland, Sheffield.
Mrs. Fred. Bland, Sheffield.
Mr. W. H. Bleckly, Warrington.
Mr. Clifford E. Bloomer, Halesowen.
Mr. J. E Alger Blyde, Sheffield.
Mrs. J. E. Alger Blyde, Sheffield.
Mr. Arthur Booth, Leeds.
Mrs. Arthur Booth, Leeds.
Mr. William Booth, Leeds.
Mrs. William Booth, Leeds.
Mr. James Bott, Eaglescliffe, Durham.
Mrs. James Bott, Eaglescliffe, Durham.
Mr. George R. H. Bowden, Gloucester.
Mrs. George R. H. Bowden, Gloucester.
Mr. Harold Bowman, Preston.
Mr. Harold E. Bowman, Preston.
Mr. Cyrus Braby, London.
Mrs. Cyrus Braby, London.
Mr. R. Watson Bradley, Stirlingshire.
Mrs. R. Watson Bradley, Stirlingshire.
Mr. V. M. Braschi, Mexico City, Mex.
Mrs. V. M. Braschi, Mexico City, Mex.
Miss Briggs, Eccles.
Mr. William Bright, Gowerton.
Mr. Fred. S. Brightmore, Doncaster.
Mrs. M. M. M. Brightmore, Doncaster.
Mr. Wallace Broad, Shanghai.
Miss Broad, Shanghai.
Mr. B. J. Broadway, Birmingham.
Miss Broadway, Birmingham.
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Mrs. J. E. Brooks, Leeds.
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Mrs. A. M. Buchanan, Cardiff.
Mr. H. A. Bueck, Berlin.
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Mrs. C. Bullock, Manchester.
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Mrs. A. M. Butchart, Barrow-in-Furness.
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Mr. Edmund Butler, Leeds.
Mrs. Edmund Butler, Leeds.
Miss Butler, Leeds.
Mr. Harold Butler, Halifax, N. S.
Mrs. Harold Butler, Halifax, N. S.
Mr. Thomas Campbell, Glasgow.
Mrs. Thomas Campbell, Glasgow.
Mr. Alfred Campion, Glasgow.
Mr. Athol John Capron, Sheffield.
Mr. Charles George Carlisle, Sheffield.
Mrs. Charles George Carlisle, Sheffield.
Mr. Edward Carlisle, Workington.
Mrs. Edward Carlisle, Workington.
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Mrs. A. E. Carlton, New York, N. Y.
Mr. David Carnegie, Sheffield.
Mr. Charles George Carson, London.
Mr. Charles Catlett, Staunton, Va.
Mrs. Charles Catlett, Staunton, Va.
Miss Catlett, Staunton, Va.
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Mrs. Geo. Cawley, London.
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Mrs. H. S. Chamberlain, Chattanooga.
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Mrs. W. F. Cheesewright, London.
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Mrs. William T. Cheesman, Nottingham.
Mr. Frederick Cleeves, London.
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Mrs. Albert Ladd Colby, New York.
Mr. David Colville, Motherwell.
Mrs. David Colville, Motherwell.
Mr. E. G. Constantine, Motherwell.
Mrs. E. G. Constantine, Motherwell.
Mr. Joseph Cook, Alfreton.
Mr. Thomas Cook, Sheffield.
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Mrs. Arthur Cooper, Middlesbrough.
Mr. Joseph Cooper, Jarrow-on-Tyne.
Mr. Edwin Cottam, Cardiff.
Mrs. Edwin Cottam, Cardiff.
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Mrs. T. F. Craddock, London.
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Mrs. T. I. Crane, Philadelphia, Pa.
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Mrs. T. Crompton, Ashton, near Wigan.
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Mrs. Walter Crooke, Jr., Middlesbrough.
Mr. J. F. L. Crosland, Hale, Cheshire.
Mr. John Crum, Stoke-on-Trent.
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Miss Cunningham, Glasgow.
Mr. Jos. C. Custodis, London.
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Mr. John Henry Darby, Brymbo.
Mrs. John Henry Darby, Brymbo.
Mr Frank Davenport, Eccles.
Miss Davenport, Eccles.
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Mrs. George M. Davidson, Chicago, Ill.
Mr. John Cecil Davies, Gowerton.
Mr. John Prosser Davies, Dowlais.
Mrs. J. P. Davies, Dowlais.
Mr. W. H. Davies, Stoke-on-Trent.
Miss Delane, Llangennech.
Mr. Carl Dellwik, London.
Mrs. Carl Dellwik, London.
Mr. T. J. Denny, Paris, France.
Mrs. T. J. Denny, Paris, France.
His Grace, The Duke of Devonshire.
Mr. John Henry Dewhurst, Sheffield.
Mrs. John Henry Dewhurst, Sheffield.
Mr. Edward Dickinson, Sheffield.
Mrs. Edward Dickinson, Sheffield.
Mr. John Dickson, Grosmont.
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Mrs. Walter Dixon, Glasgow.
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Mr. W. H. Donner, Pittsburg, Pa.

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Mr. Nicholas Downing, Stockton-Tees.
Mrs. Nicholas Downing, Stockton-Tees.
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Mr. J. T. Dunn, Newcastle-on-Tyne.
Mr. James Dunnachie, Coatbridge, N. B.
Mrs. James Dunnachie, Coatbridge, N. B.
Mr. Theodore Dwight, New York, N. Y.
Miss M. A. Eames, Merthyr Tydfil.
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Mrs. David Evans, Llanelly.
Mr. David Evans, Sheffield.
Mrs. David Evans, Sheffield.
Mr. David Llewellin Evans, Cardiff.
Mr. Henry M. G. Evans, Llangennech.
Mrs. Henry M. G. Evans, Llangennech.
Mr. John Evans, Dowlais.
Mrs. John Evans, Dowlais.
Mr. William Evans, Merthyr Tydfil.
Miss Evans, Cardiff.
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Mrs. George J. Eveson, Birmingham.
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Mrs. B. F. Fackenthal, Easton, Pa.
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Mr. William Farrar, Sheffield.
Mrs. William Farrar, Sheffield.
Mr. Finlay Finlayson, Airdrie.
Mr. Ambrose Firth, Sheffield.
Mr. J. E. Fisher, Westbury, Wilts.
Mrs. J. E. Fisher, Westbury, Wilts.
Mr. David Flather, Sheffield.
Mrs. David Flather, Sheffield.
Mr. W. T. Flather, Sheffield.
Mrs. W. T. Flather, Sheffield.
Mr. H. P. Fogh, Seattle, Wash.
Mrs. H. P. Fogh, Seattle, Wash.
Mr. Paul R. Forbes, Boston, Mass.
Mr. Henry Foster, Monmouthshire.
Mr. William James Foster, Walsall.
Mrs. William James Foster, Walsall.
Miss Foster, Pontypool.
Mr. A. A. Fownes, South Shields.
Mrs. A. A. Fownes, South Shields.
Mr. Alfred Fox, Arequipa, Peru.
Mrs. Alfred Fox, Arequipa, Peru.
Mr. Benjamin Freeborough, Sheffield.
Mrs. Benjamin Freeborough, Sheffield.
Mr. W. E. Freir, London.
Mrs. W. E. Freir, London.
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Excursions and Entertainments.

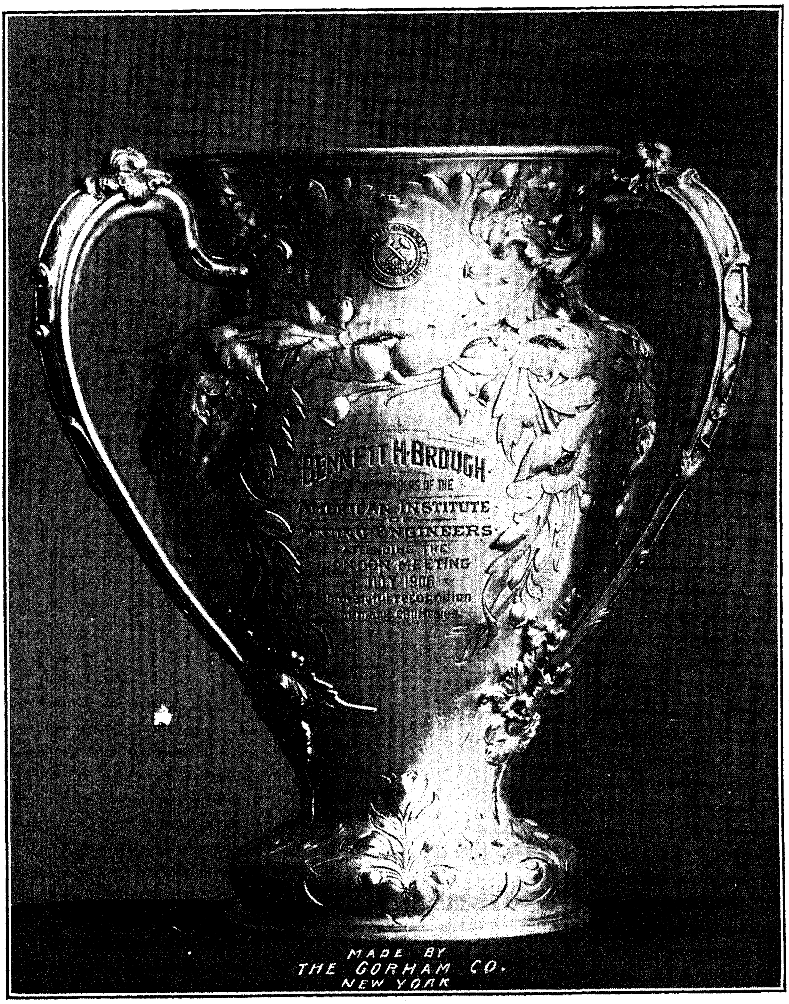
An extended account of the various excursions and entertainments was published in *Bi-Monthly Bulletin*, No. 12, November, 1906 (pp. 860–907), also in the specially bound pamphlet edition of the Proceedings of the London Meeting.

Testimonials.

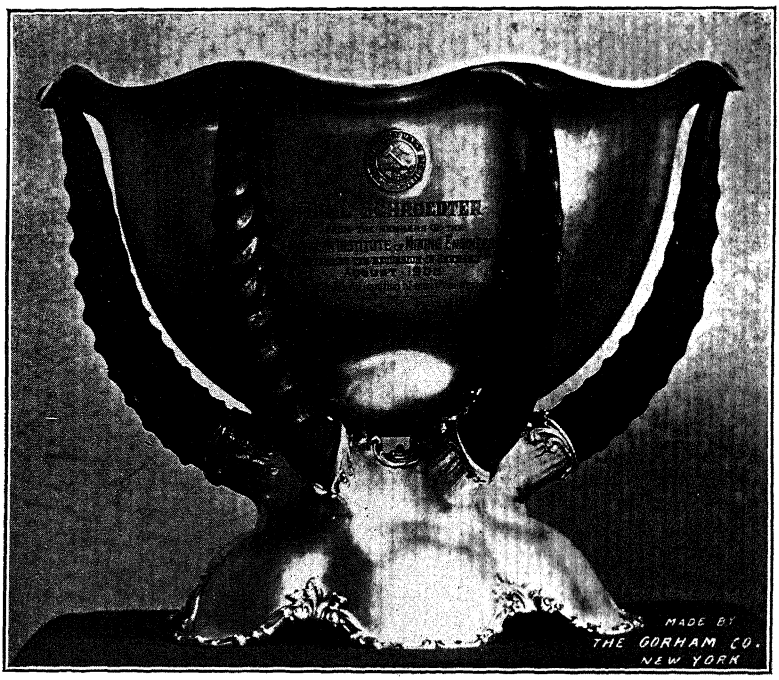
Engraved certificates, expressing the thanks of the Institute, and signed by the President, the Secretary, and four past-Presidents, were subsequently sent to individuals, firms, corporations and institutions specially active in the entertainment

of visiting members and guests in England, Scotland, Wales and Germany. The extent and cordiality of this welcome may be inferred from the fact that about 500 such official acknowledgments were required.

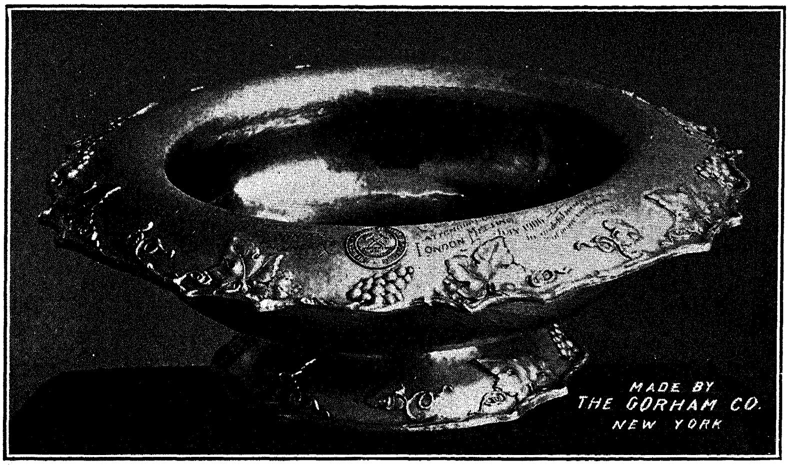
In addition to these official certificates, special testimonials were presented by the members of the visiting party to Secretary Brough, Mr. Sidney, the Stewards of the British excursion, and Drs. Schroedter and Weiskopf, and Mr. Lemke, of the German Reception Committee. These testimonials are shown in the accompanying illustrations.



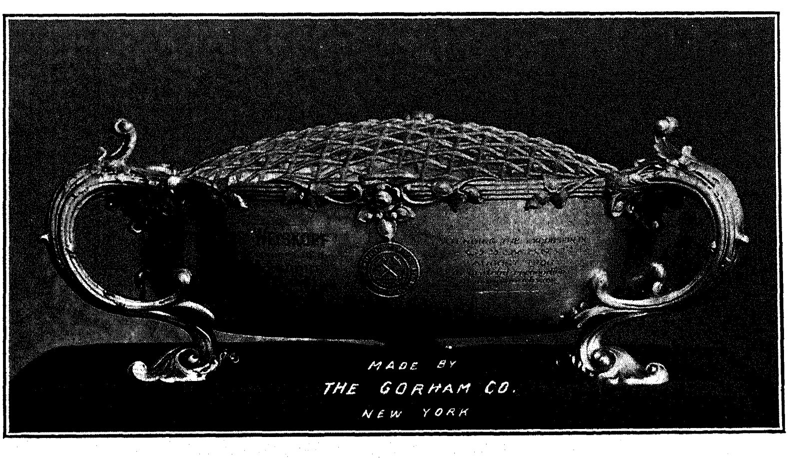
SILVER LOVING-CUP, PRESENTED TO PROFESSOR BENNETT H. BROUGH.



SILVER PUNCH-BOWL, PRESENTED TO DR. E. SCHROEDTER.



Silver Fruit-Dish, Presented to Mr. L. P. Sidney.



Silver Flower-Dish, Presented to Dr. A. Weiskopf.

GOLD WATCH-FOBS, PRESENTED TO PROF. H. BAUERMAN, MESSRS. W. F. CHEESEWRIGHT, D. A. LOUIS, A. C. MEYJES, L. PENDRED, HUBERT S. THOMAS AND R. LEMKE.

PAPERS.

Fine Grinding of Ore by Tube-Mills, and Cyaniding at El Oro, Mexico.

BY G. CAETANI, ROME, ITALY, AND E. BURT, EL ORO, MEXICO.

(Bethlehem Meeting, February, 1906.)

CONTENTS.

I. Introduction.

We owe to the courtesy of Mr. R. M. Raymond, Manager of the El Oro Mining & Railway Co., Ltd., the permission of publishing in this paper the results of a series of experiments and tests made with a view to determining the economical limit to which the fine grinding of the ore by tube-mills could be carried on.

For the sake of clearness, the following definitions are given of certain terms used in this paper, as well as of the units and screens in the experiments.

Slime: In a general way "slime" is that mill-product, of which from 90 to 95 per cent. will pass a 200-mesh screen (0.067 mm. aperture), and which is treated as one class of material in the El Oro cyanide-plants. As a screen-product, slime constitutes all material that will pass a 200-mesh screen.

In sizing, the following products are distinguished:—

Coarse Sand: The material remaining on a 100-mesh (0.111-mm. aperture) screen.

Fine Sand: That passing through 100-mesh and remaining on a 200-mesh (0.067-mm. aperture) screen.

Slime: That passing through 200-mesh. *Fines*, which pass through 200-mesh and settle in 30 seconds, and *silt*, which does not settle in 1 minute.

Sand-index: The number proportional to its total fineness considered from the economical point of view, as explained on page 13.

Absolute extraction: The percentage of values actually brought into solution.

Actual extraction: The percentage of values actually brought to the zinc-boxes.

Units: Dollars, U. S. Currency; pounds; and short tons (2,000 lb.). Gold is valued at $20.67, and silver at $0.50 per ounce.

Screens: In the sizing-tests the screens used were:—

Mesh (per 1 in.),	24	30	40	60	80	100	150	200	250
Aperture (mm.), .	0.740	0.515	0.424	0.250	0.159	0.111	0.090	0.067	0.058

Strengths of solution: Strengths are expressed in the percentage of KCN contained, though the sodium cyanide, which is

used exclusively at El Oro, corresponds to 125 per cent. of KCN.

II. The El Oro Mine.

The El Oro Mining & Railway Co., Ltd., operates a mine at El Oro, in the State of Mexico, about 100 miles NW. from the City of Mexico. The ore is gold-bearing quartz, containing a small amount of silver, and occurs in large ore-bodies in slate. The whole vein-bearing formation has been covered by a heavy andesite flow, in places several hundred feet thick, and, therefore, seldom do the veins outcrop. The principal vein is the "San Rafael," which strikes N. 10° W. and dips 63° towards the west. This vein averages 50 ft. in width, and has been followed for more than a mile in length.

The Ore.—The ore thus far worked in the El Oro mine is quite oxidized, and only a small amount of pyrite is met with at the present depth of the mine. The ore is treated by amalgamation on plates in the stamp-mill, and the tailings from these are cyanided.

The oxidized ore, a very hard, compact quartz, averages in value from $6 to $15 gold, and from 3 to 5 oz. of silver, per ton.

The quartz has been deposited by aqueous solutions, and, in places, shows a distinctly banded or wavy structure.

The gold and silver are scattered through the quartz in a remarkably fine state of division and, locally, in a very uniform way. Probably all the gold is native, and part of it is in the form of "rusty gold;" but it is so finely divided that seldom can "colors" be detected. The ore, crushed through 40-mesh, will not yield to amalgamation more than 18 per cent. of the gold, the remainder being so encased in the quartz that even the cyanide can dissolve only 80 per cent. of it from sand-grains 0.08 mm. in diameter.

The silver is probably partly metallic and partly in the form of silver sulphides, arsenides and antimonides. The amount of these metallic sulphides, however, is so small that the naked eye is generally unable to detect them.

III. Outline of Ore-Treatment.

1. *Mills and Cyanide-Plants.*—There are two mills, each containing 100 stamps. No. 1 was built in 1899, and No. 2 was completed but a few months ago.

In No. 1 the sand and slime are treated practically in the same way as in plant No. 2, though the material arrangement and the mechanical devices are different. The greatest difference between the two plants is, that No. 1 has no tube-mills for regrinding the sand. At present the coarse sand is brought over to plant No. 2 to be reground and treated; though in the near future two tube-mills will be added for grinding exclusively the sand from No. 1.

The ore is dumped from skips into ore-bins, crushed through gyratory and jaw-crushers, and thence conveyed to the mills by cars, or by belt-conveyors. The mills are of the standard Allis & Chalmers pattern. The ore is stamped through from 25- to 35-mesh screens, and runs over amalgamated plates, which catch from 13 to 18 per cent. of the gold and from 1 to 5 per cent. of the silver. The bullion from the retorted amalgam contains about equal parts by weight of silver and gold. The pulp from the plates is reground through tube-mills and then flows to the cyanide-plants, where it is separated into slime and sand, which are treated separately.

Fig. 1, showing the method of treatment at Mill No. 2, is self-explanatory; but, as the experiments given in this paper concern almost exclusively Mill No. 2, we shall give, regarding this plant, additional information which it was not possible to put on the diagram.

In Mill No. 2 the ore is crushed by 100 stamps, each weighing 1,140 lb., and dropping 6 in. 100 times per min.; the discharge is 2.5 in. through 25-mesh brass wire screen (0.74-mm. aperture). The duty per stamp is 4.7 tons per 24 hr. of actual work. The ratio of mill-water to ore is from 9 to 10.

Owing to the scarcity of water in the district, the mill-water, after separation from the slime, has to be pumped back to the large storage-tanks. About 340 tons of mill-water pass daily into the cyanide solutions, being carried in as moisture in the slime, which contains 53 per cent. of water, and in the sand, which contains 15 per cent. of water. The discharged slime carries 40 per cent., and the sand 15 per cent., of water; therefore, about 75 tons of weak solution have to be discharged daily.

The pulp from the plates is classified by two 4.5-ft. cones put in series with two 2-ft. cones, called "pulp-thickeners," giving

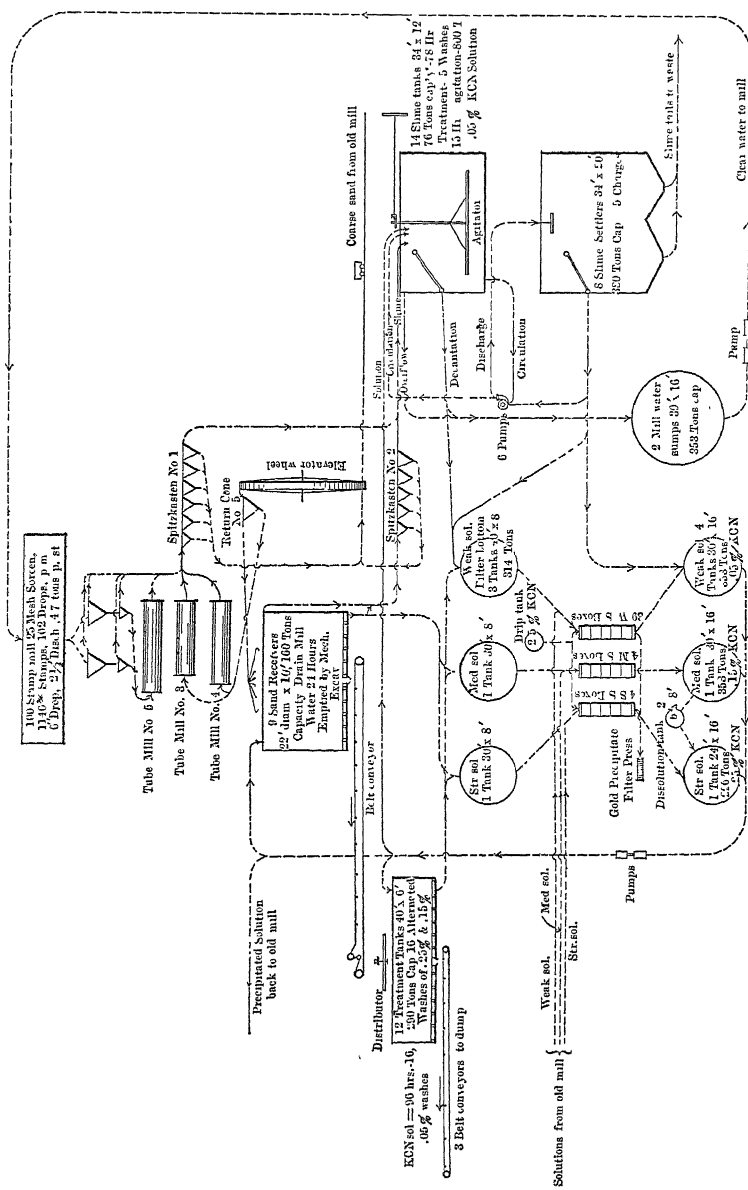


FIG. 1.—DIAGRAM OF CYANIDE TREATMENT AT PLANT NO. 2. (September, 1905.)

two classified products, namely,—an overflow containing about 81 per cent. of slime and representing 52 per cent. of the total mill-output, and a discharge containing 9 per cent. of slime.

The discharge of the pulp-thickeners is run through tube-mills, where from 40 to 50 per cent. of the sand is slimed, ac-

cording to the various conditions under which the tube-mills are working.

The pulp discharged by the tube-mills joins the overflow from the four cones and forms a final product, containing, on an average, 69 per cent. of slime. The product is then classified and handled, as shown in Figs. 1 and 2.

The final result is to separate two mill-products,—sand and slime; the former containing about 20 or 25 per cent. of slime, and representing about 35 per cent. of the rock crushed, and the latter containing about 5 or 10 per cent. of sand.

2. *Classification of Mill-Products.*—Numerous sizing-tests were made to determine the classification wrought by the cone, spitzkasten, etc., and to measure the grinding-efficiency of the tube-mills.

In order to grasp more easily the meaning of the numerous figures thus obtained, a graphic method of representation was resorted to, and, wherever possible, this method has been used to represent all the other tests and experiments.

For each sizing-test, a small diagram was drawn, the percentage of each size of sand being employed as an ordinate of the diagram, as is shown in Fig. 2. The coarser the sand, the more will the diagram swell to the left; on the other hand, a perfect slime will be represented by the dotted triangle on the right, as shown in the legend at the upper left-hand corner of Fig. 2. This illustration shows also the handling and classification of the mill-pulp from Mill No. 2. The small sand-diagrams, inserted at the point where each separation comes to an end, exhibit the transformation produced in the sand.

The method of classification shown in Fig. 2 is at present undergoing modification. In the near future the buckets of the sand-wheel will be divided into two compartments; one, for elevating the coarse sand which has to be reground in the tube-mills, and the other, for elevating the fine sand which goes to the sand-receivers. By such a method the capacity of the sand-wheel will be doubled. The spitzkasten will be cut out, and cones used exclusively for classifying. By this arrangement it is hoped to obtain a closer separation of coarse sand, fine sand and slime,—a condition which is essential for economically regrinding the coarse sand.

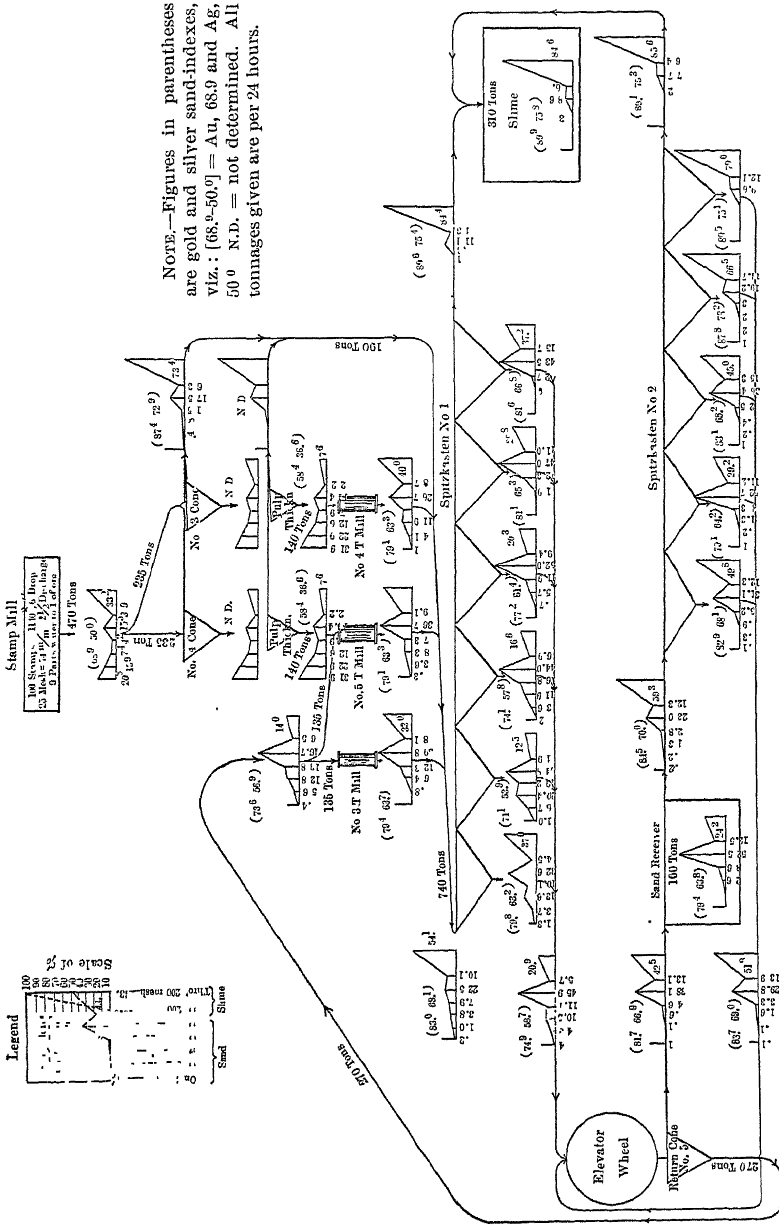


FIG. 2.—DIAGRAM OF SIZING-TESTS AT MILL NO. 2. (September 1, 1905.)

IV. RELATION BETWEEN DIAMETER OF SAND-GRAINS AND EXTRACTION.

The relation between the diameter of the sand-grains and the maximum possible gold- and silver-extraction by the cya-

nide process has been taken as the starting-point in the determination of economical limit in regrinding the sand.

For this purpose all the available data were collected, and several sizing-assay tests were made, both of the untreated pulp and of the tails. These data have been plotted in Figs.

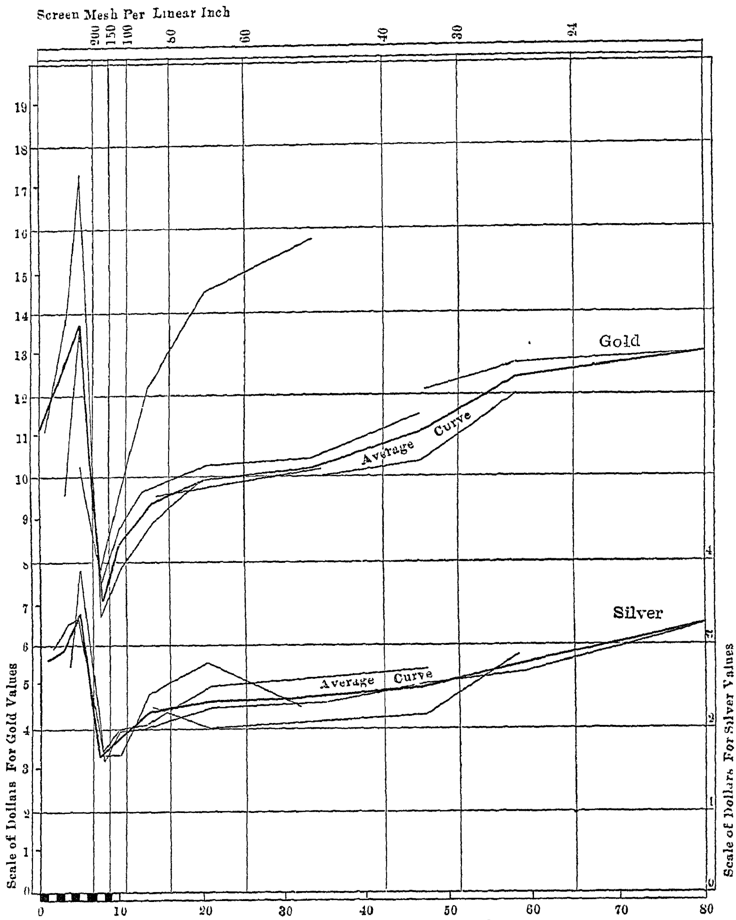


FIG. 3.—RELATION BETWEEN DIAMETER OF SAND-GRAINS AND VALUES OF HEADS.

3 and 4,[1] the former giving the gold- and silver-values of the sand-grains of a given diameter before treatment; and the

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[1] In these diagrams are included the sizing-assay tests published by Charles Butters and E. M. Hamilton in their valuable paper, "The Cyaniding of Ore at El Oro," etc., *Transactions of the Institution of Mining and Metallurgy*, October, 1904.

latter, the values of the same grains after treatment, as well as the resulting average gold- and silver-extraction.

The plotting of these diagrams was rendered possible only by the reduction of all the assay-values to a common base, namely, to an ideal assay-value of the original pulp of $10 of

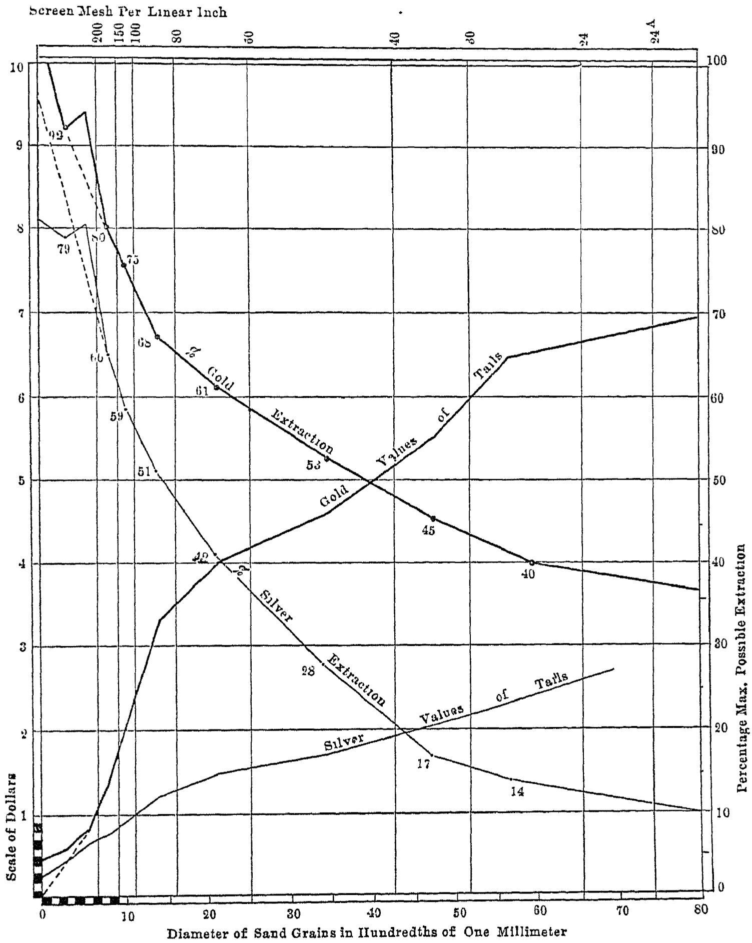


FIG. 4.—RELATION BETWEEN DIAMETER OF SAND-GRAINS, VALUES OF TAILS AND EXTRACTION.

gold and 5 oz. of silver per ton. This assumption is justified by the fact (verified by experiments) that the values of the tails are in linear proportion to the value of the heads. (A high-grade slime will give a somewhat higher extraction than a low-grade one; but the difference is small.) Therefore, all

gold-values, both of the heads and the tails, were multiplied by $10 and divided by the gold-assay value of the untreated pulp; and the silver-values were multiplied by 5 oz. and divided by the silver-value of the untreated pulp.

Examining the diagrams of the values of the heads in function of the diameter of the grains, Fig. 3, it is seen that the values of the sands decrease, in a fairly constant way, as the size of the grain decreases, until a diameter of 0.15 mm. is reached. From 0.15 to 0.08 mm. these values drop rapidly, reaching a minimum at 0.08 mm.; then they rise rapidly, being, at 0.05 mm., far above the average value of the pulp, and afterwards begin to drop again for those grains which constitute the slime. The values of the tails, Fig. 4, decrease proportionally to the diameter of the grain, but do so much more rapidly between 0.15 and 0.08 mm. To explain this anomaly, both for the heads and for the tails, the intimate structure of the ore must first be investigated.

The El Oro ore is a very hard, compact quartz, in which the values are disseminated in a remarkably fine state of division and, locally, in a very uniform way. The quartz was evidently deposited from aqueous solutions, and probably the gold and silver were precipitated at the same time as the quartz. The structure of the quartz is micro-crystalline, and the silver-minerals and the gold were precipitated between the faces of the elementary quartz crystals.

The structure of the ore, however, is remarkably compact, and therefore practically impermeable to the solutions; hence, only those minerals can be brought into solution which are on the surface of the sand-grains, or which are so encased in the quartz that at least a part of their surface is exposed. Consequently, the maximum extraction must be proportional to the surface of the ore exposed by crushing; or, in other words, must be inversely proportional to the square of the diameter of the grains. It follows that the curves representing the average gold- and silver-extraction, in relation to the diameter of the grains, ought to have the form of a parabola, and this is about what has been found by experiment, as can be verified by Fig. 4.

What we have just said about the structure of the ore suggests another consideration, namely, the breaking-off of the

metallic mineral from the surface of the sand-grains. This breaking-off will produce an impoverishment of the sand-grains, which will be so much more sensible as the diameter of the grains becomes smaller. That class of sand, the grain-diameter of which is equal to the diameter of the elementary quartz crystals constituting the ore, will present a material in which the greatest part of the metallic minerals will be subject to breaking-off from the quartz crystals; or, at least, will be most completely exposed to the dissolving-action of the cyanide.

This is what probably happens when the ore is crushed to 0.08 mm. grains, since 0.08 mm. is probably the average diameter of the elementary quartz crystals.

On the other hand, the metallic minerals are so small that they can almost never be detected by the naked eye, and, therefore, will pass easily through a 200-mesh screen; but, owing to their high specific gravity, they will hydraulically separate with the very fine sand passing 200-mesh.

These considerations explain satisfactorily the anomalies shown in all the curves plotted on Figs. 3 and 4. Perhaps other explanations will be found; but one fact remains beyond dispute, namely, that both the gold- and silver-extraction are greatly increased by crushing finer than 0.10 mm., and, as the crushing to such fineness is economically possible by means of tube-mills, it may be safely said that the economical point in regrinding is beyond the size of 0.10-mm. grains.

V. Sand-Index.

From the foregoing, the maximum possible extraction is seen to be constant for each size of sand-grains, independently of the original assay-value of the pulp.

It is, therefore, easy to understand that we can safely calculate, *a priori*, the probable gold- and silver-extraction for a class of sand when its mechanical composition is known by sizing-tests.

To do this, we simply have to multiply the percentage of sand retained on each sieve by the average percentage-extraction for that class of sand, and then to take the sum of all the products thus obtained. The probable percentage of extraction thus calculated is what we shall call throughout this paper the "index" of a given class of sand, since it represents a number

proportional to the fineness of the sand considered from the economical point of view. To be perfectly exact, we ought not to use as coefficient the percentage-extraction shown by the extraction-curves in Fig. 4; since the values of the heads are not constant for the different sizes of grain. Each coefficient ought to be modified proportionately to the ratio between the assay-values of the sand-grains to which it refers, and the assay-value of the total pulp; but in practice the sand-indexes check so closely with the extraction actually obtained, that there is no need to complicate the calculations in this way.

As an example, we give the calculation of the sand-indexes for the 281-ton lot of sand treated in test "T," Fig. 8.

TABLE I.—*Sand-Indexes of 281-Tons Treated by Test "T."*

1	Screens used ; mesh	On 40.	On 60.	On 80	On 100.	On 150	On 200.	Thro' 200.	Total.
2	Per cent. sand retained.......	0	0.2	1.0	3.6	35.4	16.8	43.0	100
3	Av. diameter of grains, mm.	0.47	0.33	0.26	0.14	0.10	0.09	0.04	....
4	Per cent. probable gold extraction.........................	45	53	61	68	75	80	92	....
5	Per cent. probable silver extraction.........................	17	28	42	51	59	66	79	....
6	Product of per cent. sand by per cent. gold extraction..	0	0.10	0.61	2.44	26.60	13.44	39.60	82.8
7	Product of per cent. sand by per cent. silver extraction.	0	0.05	0.42	1.84	20.90	11.10	34.00	68.3

The calculated extractions, or sand-indexes, are :—gold, 82.8 ; silver, 68.3 ; while the actual extractions in test "T" were :— gold, 82.9 ; silver, 69.5. For brevity, we shall represent the gold index, Au. 82.8, and the silver index, Ag. 68.3.

The sand-indexes have a great importance in the determination of the economical efficiency of the tube-mills.

Every time a sand passes through a tube-mill its composition becomes finer; the ground product will yield a larger proportion of its values to the cyanide treatment, or, in other words, a certain percentage of the gold and of the silver will have been freed by the action of grinding. This percentage represents the gross profits obtained by the work done. The economical point of regrinding will be reached when the difference between the total values freed and the total cost of regrinding will be but a few cents. This, however, does not mean that it

would pay to slime all the sand, since the cost of grinding increases rapidly with the fineness of the sand to be reground.

Fine grinding will increase the total ratio of slime to sand, and, therefore, would apparently tend to diminish the cost of treatment for the slime and increase that for the sand.

This result, however, does not correspond to the practice, as the tonnage of sand to be treated on the 200-ton sand-treatment plant will be kept constant by sending the coarse sand of the old mill to the new plant to be reground and treated. Besides this, there are about 500,000 tons of old tailings, containing about $4.60 in gold and 2 oz. of silver per ton, which some day will have to be reground and treated.

VI. Tube-Mills.

At present there are three tube-mills in use at the new plant, and two additional ones will soon be added for grinding the sand of the old mill. The mills were constructed by the Fried. Krupp Aktiengesellschaft Grusonwerk, of Magdeburg, Germany, and the principal data concerning them are as follows:—

Mill	No. 3.	No. 4.	No. 5.
Internal diameter of drum	47 in.	59 in.	59 in.
Internal length of drum	19 ft. 6 in.	23 ft.	26 ft.
Revolutions per minute	32	27.5	28.5
Pounds of pebbles in mill when half full	11,700	21,800	24,700
Capacity per 24 hr., tons	100–120	130–150	150–180
Cost of mill	$2,940.	$3,850.	$4,750.

Bulk of pebbles = 100 lb. per cu. ft.

The tube-mills are set on concrete foundations and held in place by anchor-bolts. Experience thus far at El Oro indicates that the foundations cannot be made too solid. The mills, when in motion and heavily loaded with pebbles, bear very heavily on the foundations, which, owing to the mechanical arrangement of the mills, cannot be made on the top as wide as is desirable. The cast-iron bed-plates have a tendency to dig into the cement, and it is advisable to enlarge the bearing-surface of the bed-plates by placing larger cast-iron plates under them. A piece of rubber belt placed below these plates will be

found to be advantageous. The vibrations produced by the spur-gears are violent, and the anchor-bolts easily crystallize and break; therefore, pockets are advisable. The driving-pinion ought to revolve in such direction as to lift the gearing of the tube-mill, and press down on its own bearing. On the contrary, it is continuously pulling on the anchor-bolts of the boxes, thus tending to loosen the nuts and to break the bolts.

Notwithstanding this objection, we can safely declare that, in our opinion, the tube-mills are the best machines tried at El Oro for the fine grinding of very hard quartz sand.

Of the three tube-mills in use, No. 3, the smallest, has given most satisfaction and least trouble. Its capacity is not much below that of the other two; but the cost of running it is much smaller. The main advantage is that the mill is comparatively light and compact, and, therefore, seldom gets out of order. The shut-down of one mill represents a great loss in money; hence, reliability is more important than capacity.

The liners are kept in place by counter-sunk bolts passing through the shell. When a liner gets worn out the bolts begin to leak, and thus give notice that that special liner needs to be replaced. Such leakage, which is a useful indication regarding the liners, ought, however, to be avoided for the bolts holding the head-plates; because the sand leaking from behind the bolts will move by centrifugal force towards the driving-gear, and cause a rapid wearing of the cogs.

The three tube-mills are placed parallel to each other, and are driven by belts from the line-shaft of the stamp-mill. It would be of great advantage to have positive clutches on the pulleys of the line-shaft, as any repairs necessary to the belts or to the driving-pulleys of the tube-mills require the shut-down of the whole stamp-mill.

The pulp from the stamps is, as previously mentioned, classified into coarse sand, and an overflow containing 81 per cent. slime and some fine sand. The coarse sand can be sent to the head of any of the three mills, or divided among them. The insufficiently-ground sand leaving the mills is elevated, separated from the fine sand and slime, and can be likewise returned to any of the three mills, or divided among them. Figs. 5 and 6 give two diagrams of the combination generally used in grinding and regrinding the sand.

1. *Grinding-Efficiency.*—The tube-mills have been working on sand of different coarseness. In May the stamps were crushing through 40-mesh, in June through 30-mesh, and latterly through 25-mesh screen.

About 150 sizing-tests were made of the heads and tails. It is very hard to discuss the results of such work, especially because the variable factors in the operation of grinding vary

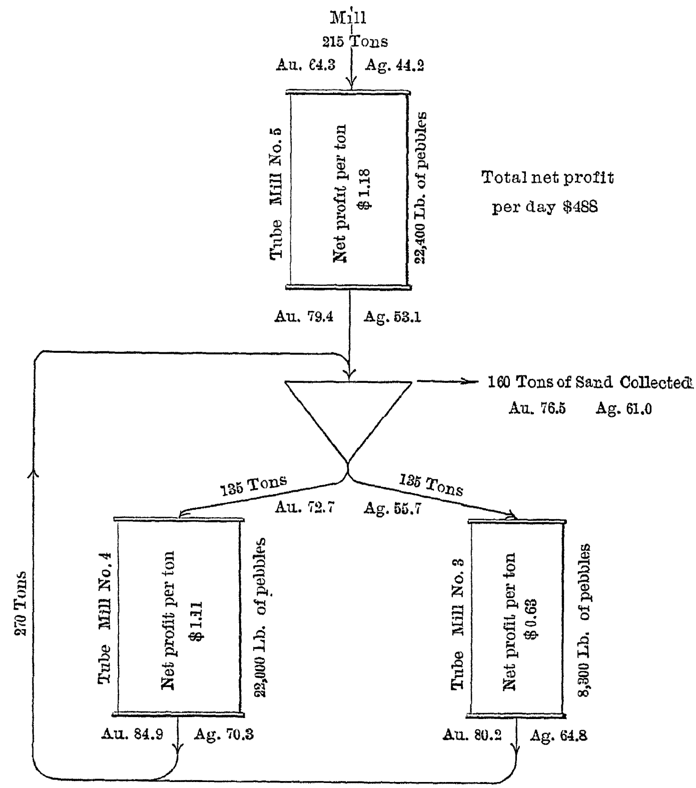


FIG. 5.—DIAGRAM SHOWING THE METHOD OF GRINDING AND REGRINDING SAND IN TEST "O." (August 13, 1905.)

independently of each other. In an experimental plant it would be possible to vary one of these factors alone (such as quantity of pebbles, rate of feed, ratio of water to sand, etc.), and note the relation of all the other factors to the variable one; but in experimenting with a working-plant this is not practicable.

For the same reason it was not feasible to represent graphi-

cally all the results obtained; hence we have been obliged to tabulate them.

The understanding of the tables is greatly facilitated by the use of the sand-indexes, by means of which we can represent with one figure (dollars and cents) the work wrought by a tube-mill under given conditions. Without the sand-indexes we

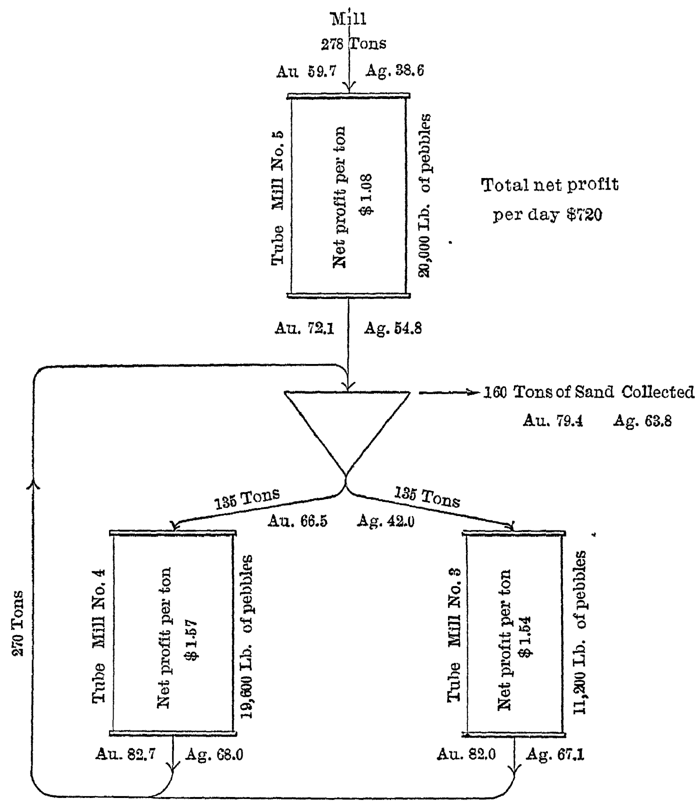


FIG. 6.—DIAGRAM SHOWING THE METHOD OF GRINDING AND REGRINDING SAND IN TEST "R." (August 17, 1905.)

would have to study the relation of 14 numbers among themselves.

Economically, our only interest in sand is to know the value, in dollars and cents, that can be extracted from it. Some grinding-machinery may produce much slime, and some comparatively coarse sand; other machines may be poor slimers, but produce a large percentage of very fine sand. By means of the sand-indexes we realize immediately which of the two products

is the finer, considered from an economical point of view, while at times it would be impossible to do so by simply looking at the figures given by the sizing-tests.

Tables II. to V. give the results of a number of tests made on the three tube-mills. The factors which vary and influence the efficiency of the tube-mills are the coarseness of the heads, the rate of flow, and the amount of pebbles in the mill. The other factors, namely, the character of the ore, the ratio of sand to water, size of pebbles, etc., may be considered to have remained constant throughout the tests.

The moisture of the pulp fed to the mills averaged from 50 to 75 per cent.

In Tables II. to V. the coarseness of the sand is represented by the gold- and silver-indexes; the useful work done by the tube-mills is represented by the increased extraction produced, by the gross profit per ton ground, or by the net profit obtained per ton, or per 24 hr., of continuous run.

In Table V. are given the sizing-tests of a number of sands of different sand-indexes. The reader, wishing to know the composition of a sand having, say, a gold sand-index of 84.9 per cent., can refer to this table, and get a fairly exact idea of what would be a sizing-test on such class of sand.

Naturally, there is an endless number of sands of a given sand-index, but of different composition; although, with the present system of classifiers in use, such differences are always small.

The reader can also refer to Fig. 2, from which are taken several of the sizing-tests given in the Tables II. to V.

It is difficult to discuss completely the results tabulated in Tables II. to V., owing to the reasons already mentioned. However, we can safely assert, that:—

(1) *The efficiency increases proportionally to the amount of pebbles contained in the mill*, as may be seen:—

In Table II. by comparing Test No. 2 with Test No. 6.

In Table II. by comparing Test No. 5 with Test Nos. 9 and 12.

In Table III. by comparing Test No. 2 with Test No. 8.

In Table III. by comparing Test No. 6 with Test No. 9.

In Table IV. by comparing Test No. 1 with Test No. 4.

In Table IV. by comparing Test No. 2 with Test Nos. 5 and 6.

TABLE II.—*Showing the Efficiency of Tube-Mill No. 3.*

No.	Tons per 24 Hr.	Pounds of Pebbles in Tube-Mill.	Heads.			Tails.			Increased Extraction.		Gross Profit per Ton.		Total Net Profit.	
			Au.	Ag.	Per Cent. Slime.	Au.	Ag.	Per Cent. Slime.	Au.	Ag.	Au.	Ag.	Per Ton	Per 24 Hr.
			Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent.	$	$	$	$
1	130	....	63.8	44.0	5.7	81.7	66.2	44.5	17.9	22.2	1.43	0.38	1.47	191
2	130	12,000?	64.8	45.7	7.3	82.4	67.3	47.0	17.6	21.6	1.41	0.37	1.44	187
3	135	11.200	66.5	42.0	8.8	82.0	67.1	50.6	15.5	25.1	1.24	0.43	1.33	180
4	100?	.. .. .	63.4	42.9	7.1	83.4	66.7	46.0	20.0	24.6	1.60	0.42	1.68	168
5	84	12,000?	64.1	44.0	7.5	86.0	71.3	62.0	21.9	27.3	1.75	0.46	1.87	157
6	130	8,000?	66.4	47.4	7.9	79.3	63.6	34.9	12.9	16.2	1.03	0.27	0.96	125
7	69	12,400?	65.0	45.2	8.2	85.9	71.1	61.3	20.9	25.9	1.67	0.44	1.77	123
8	100?	7,500?	63.6	46.2	10.3	79.4	63.7	33.4	15.8	17.5	1.26	0.30	1.22	122
9	93	10,000	66.4	47.3	7.8	82.1	66.9	44.5	15.7	19.6	1.26	0.33	1.25	116
10	250	11,000	67.1	47.5	11.0	73.5	56.5	22.0	6.4	9.0	0.51	0.15	0.32	80
11	75	....	66.9	48.1	7.9	81.3	65.9	42.4	14.4	17.8	1.15	0.30	1.11	83
12	75	9,000?	67.5	50.0	8.7	79.5	64.0	40.0	12.0	14.0	0.96	0.24	0.86	64
13	140	8,300	72.7	55.7	13.3	80.2	64.8	39.0	7.5	9.1	0.60	0.15	0.41	67
14	135	11,000	71.5	54.5	12.3	79.0	64.0	38.0	7.5	9.5	0.60	0.16	0.42	57
15	135	9,500	73.6	56.9	14.0	79.4	63.4	33.0	5.8	6.8	0.46	0.11	0.23	31
16	135	10,900	73.3	56.5	15.6	78.5	63.0	32.2	5.2	6.5	0.42	0.11	0.19	26
17	170	11,000	76.7	61.6	20.3	80.0	64.7	37.0	3.3	3.1	0.26	0.05	0.03	-5

TABLE III.—*Showing the Efficiency of Tube-Mill No. 4.*

No.	Tons per 24 Hr.	Pounds of Pebbles in Tube-Mill.	Heads.			Tails.			Increased Extraction.		Gross Profit per Ton.		Total Net Profit.	
			Au.	Ag.	Per Cent Slime.	Au.	Ag	Per Cent. Slime.	Au	Ag.	Au.	Ag.	Per Ton.	Per 24 Hr.
			Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	$	$	$	$
1	200	.. ..	64.6	44.4	8.4	83.4	68.6	51.0	18.8	24.2	1.44	0.41	1.48	296
2	150	22,000?	64.2	44.1	8.9	83.8	69.0	50.0	19.6	24.9	1.57	0.42	1.62	243
3	135	18,600	58.4	36.6	7.6	79.1	63.3	40.0	20.7	26.7	1.65	0.45	1.73	234
4	270	19,500	65.2	45.0	10.0	75.5	61.4	30.2	10.3	16.4	0.82	0.28	0.73	197
5	97	....	63.4	43.1	4.8	86.1	71.2	64.3	22.7	28.1	1.82	0.48	1.93	187
6	117	18,000?	66.2	46.9	9.8	85.5	71.1	60.8	19.3	24.2	1.54	0.41	1.58	185
7	135	19,600	66.6	42.0	8.8	82.7	68.0	50.0	16.1	26.0	1.29	0.44	1.36	184
8	139	19,400	64.7	43.7	16.3	81.0	66.0	44.4	16.3	22.3	1.30	0.38	1.31	182
9	111	16,000	63.0	42.7	4.9	80.1	64.6	38.0	17.1	21.9	1.37	0.37	1.37	152
10	135	22,000	72.7	55.7	13.3	84.5	70.3	56.5	11.8	14.6	0.94	0.25	0.82	111
11	200	....	70-5	53.3	9.1	76.8	60.8	21.5	6.3	7.5	0.50	0.13	0.26	52

TABLE IV.—*Showing the Efficiency of Tube-Mill No. 5.*

No.	Tons per 24 Hr.	Pounds of Pebbles in Tube-Mill.	Heads.			Tails.			Increased Extraction.		Gross Profit per Ton.		Total Net Profit.	
			Au.	Ag.	Per Cent. Slime.	Au.	Ag.	Per Cent. Slime.	Au.	Ag.	Au.	Ag.	Per Ton.	Per 24 Hr.
			Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	$	$	$	$
1	215	26,800	66.3	45.6	13.5	84.0	69.0	53.5	17.7	23.4	1.41	0.40	1.41	303
2	274	22,500	66.0	46.7	7.6	79.1	63.3	35.1	13.1	16.4	1.05	0.28	0.93	255
3	139	19,900	58.9	35.4	7.6	79.4	64.4	41.2	20.5	29.0	1.64	0.49	1.73	241
4	215	22,400	64.3	44.2	9.0	79.4	63.1	36.8	15.1	18.9	1.21	0.31	1.12	241
5	278	20,000	59.7	38.6	9.6	72.1	54.8	22.8	12.4	16.2	0.99	0.27	0.86	239
6	274	19,500	66.9	49.5	16.3	76.2	60.9	31.6	9.3	11.4	0.74	0.19	0.53	145
7*	72	22,000?	62.9	42.5	8.0	87.1	72.6	70.0	24.2	30.1	1.94	0.51	1.91	138
8*	52	22,000?	68.3	45.0	10.0	85.4	71.0	61.0	17.1	26.0	1.37	0.44	1.00	52
9*	24	22,000?	65.7	47.2	12.0	89.5	75.7	83.4	23.8	28.5	1.90	0.48	1.31	31

* Test by E. M. Hamilton, *Transactions of the Institute of Mining and Metallurgy*, October, 1904, pp. 13, 16, 22.

TABLE V.—*Sizing-Tests of Sands of Different Sand-Indexes.*

Sand-Indices.		Sample from Where Taken.	On						Through
Au.	Ag.		40	60	80	100	150	200	200
58.4	36.6	Discharge pulp thickeners.	31.9	22.9	12 6	7 9	14.4	2.2	7.6
63 4	43.1	Head Tube-Mill No. 4	7.5	28.0	24.5	12.0	20.3	2.9	4.8
65.5	48.4	Head Tube-Mill No. 4	1.8	14.2	21.0	15.2	27.0	5.7	10.0
67.5	50.0	Head Tube-Mill No. 3	1.9	14.9	26.0	15 4	27.4	4.8	8.7
69.2	51.6	Head Tube-Mill No. 4	1.4	12.8	22.7	16.3	26.7	5.4	13.6
71.5	54.5	Head Tube-Mill No. 4	1.1	8.6	14.3	16.0	42.1	5.0	12.3
72.8	56.1	Head Tube-Mill No. 4	......	0.7	9.8	14.5	51.0	9 3	14.3
74.1	57.8	3d plug of spitz No. 1	0.2	3.6	11.9	16.8	44.0	6.4	16.6
76.5	61.0	Sand collected	....	0.1	2.9	8.6	52.5	12.5	24.2
78.5	63.0	Tails Tube-Mill No. 3	0 1	1.1	7.8	10,1	39.2	9.5	32.2
80.1	64.6	Tails Tube-Mill No. 4	0.1	1.7	7.3	11 0	30.8	11 4	38.0
81.3	65.9	Tails Tube-Mill No. 3	......	1 3	5.9	7.1	31.5	11 8	42.4
82 4	67.3	Tails Tube-Mill No. 3	....	1.2	2.8	5.4	34.2	9.3	47.0
83.4	68.6	Tails Tube-Mill No. 4	....	0.2	2.3	4.5	27.8	13.8	51.0
84.4	69.7	Tails Tube-Mill No. 4	...	0.2	1.8	4.1	26.1	12.4	55.4
84.9	70.3	Tails Tube-Mill No. 4	..	0.1	0.8	3.8	25.7	13 2	56.5
87.8	73.2	4th plug spitz No. 2	....	....	0.5	0.3	19.2	14 7	66.5
89.9	75.8	Slime collected.	...	.	....	0.3	8.6	6.9	84.6
91.3	78.0	Slime collected	....	..	....	......	....	5.0	95.0

(2) *The efficiency increases with the coarseness of the sand fed to the mill*, as may be seen:—

On Table II. by comparing Test No. 3 with Test No. 14.

On Table II. by comparing Test No. 13 with Test No. 17.

On Table III. by comparing Test No. 3 with Test No. 7.

On Table III. by comparing Test No. 1 with Test No. 11?

On Table III. by comparing Test No. 2 with Test No. 10.

On Table IV. by comparing Test No. 5 with Test No. 6.

(3) *The efficiency decreases proportionally to the rate of feed*, as may be seen:—

On Table II. by comparing Test No. 2 with Test No. 5.

On Table II. by comparing Test No. 3 with Test No. 10.

On Table III. by comparing Test No. 5 with Test No. 1.

On Table III. by comparing Test No. 1 with Test No. 2.

On Table IV. by comparing Test No. 2 with Test No. 9.

On Table IV. by comparing Test No. 3 with Test No. 5.

Regarding the net profit per day given by a mill, it varies with the variation of the three factors just mentioned. It is difficult to say under which conditions the mills are doing the most satisfactory work, since it greatly depends upon the main object of a plant, whether it is to produce a large amount of money or to obtain a high efficiency.

Regarding high efficiency as the main object, it is important not to try to get a very fine product at the first operation by

reducing the feed below 3 tons an hour. It is far better somewhat to overcrowd the mills, to separate the slime and fine sand, and to return the insufficiently-ground sand to the mills. The returning sand can be classified, by means of cones, to any desired degree of fineness or coarseness.

Let us now suppose the coarse sand from the stamps to be ground or reground by two tube-mills placed in series, yielding a final product which, classified, will give 69 per cent. of slime and 31 per cent. of sand; 89 per cent. of the sand will pass 100-mesh, and 53 per cent. will remain on 150-mesh (Table II., Test No. 17). If we run this sand through a tube-mill the increased extraction will hardly pay for the cost of regrinding.

Suppose, then, we take the same coarse sand from the stamps and feed it very slowly to a tube-mill so as to obtain a very fine product. We shall have the mill doing two distinct things, namely, first, breaking up the coarse sand-grains to 150-mesh grains, and second, grinding such small grains to finer ones.

Now, we know that the second operation does not give a profit; and, therefore, it is more profitable to overcrowd the mill and get the fine sand through as quick as possible, than to separate the fine sand and slime and to return to the mill only that sand which it is profitable to grind.

This does not mean, however, that the El Oro ore cannot be ground economically finer than through a 100-mesh screen.

With the present system of classification by spitzkasten and cones, the fine return sand cannot be fed to the mills with less than about 70 per cent. of moisture, and fine sand needs to be ground in the state of a thick pulp, while the coarse sand can be ground in a state of greater dilution.

We have seen, previously, that the value of the tails drops rapidly for sands which will pass a 150-mesh sieve, and, therefore, it is most important (and probably will prove to be economical) to grind beyond that limit.

2. *Pebble-Consumption.*—The pebbles used at El Oro are the so-called "Danish" pebbles, and cost $36.80 per ton brought to the mill.

Tube-mill No. 3 shows an average consumption of 3.7 lb. of pebbles per ton of sand passing through. Tube-mill No. 5 shows a consumption of 3.9 lb., and No. 4 of 9.2 lb. The average consumption for the last 4 months has been 5 lb. per ton.

The reason No. 4 consumed more than twice as much as the other two mills is, that the fine sand, returning to be reground, was chiefly fed to this mill.

3. *Liners.*—The former liners came from Germany, but at present all the liners are made in the foundry of the El Oro Mining & Railway Co., Ltd. These last as long as the imported ones, and cost about one-half as much.

The consumption of liners is about 1.2 lb. per ton of sand ground, but accurate figures are not available at present.

Table VI. shows the wear of one set of Krupp liners. It is interesting to note how much greater is the rate of wearing near the feed-end of the mill, indicating that the greater part of the work is done in the first half of the mill.

TABLE VI.—*Consumption of Liners in Tube-Mill No. 3.*

Date. 1905.	Number of Liners Changed in the				Head-Plates.	Sundry.
	1st Row.	2d Row.	3d Row.	4th Row.		
May 3	Mill	started	with	new set	of	liners.
June 29	1	......	......	......	......	......
July 3	2	......	......	......	......	......
July 6	2	......	......	......	......	......
July 12	5	2	......	......	......	1 manhole.
July 13	......	2	......	......	......	......
July 14	......	5	......	......	1	1 screw.
July 19	......	......	......	4	......	......
July 25	3	4	4	......	......	......
July 30	......	......	1	......	4	1 screw.
August 15	......	......	3	......	......	......
August 16	......	......	2	6	1	......
Total	13	13	10	10	6	......

NOTE.—On August 16th, head- and tail-plates were practically worn out.

Two tube-mills have been ordered for grinding the sand of Mill No. 1; these will be provided with silex liners.

4. *Cost of Grinding.*—The cost of grinding by tube-mills has been calculated per ton of sand actually passing through the tube-mills during the last 4 months' run. The costs of grinding recorded in Tables IX. and X. are given per ton of sand and per ton of slime treated, and, therefore, differ considerably from those in Tables VII. and VIII.

During the month of August, 5,170 tons of sand were run through the tube-mills, out of the 10,773 tons crushed by the

stamps, and 72 per cent. of these 5,170 tons were returned to the tube-mills to be reground. The total amount of sand ground was 8,890 tons.

TABLE VII.—*Cost of Grinding and Regrinding by Tube-Mills Per Ton of Sand Actually Passing through the Mills.*

Items.	Tube-Mill No. 5.	Tube-Mill No. 4.	Tube-Mill No. 3.
Depreciation of capital (3 years)	$0.036	$0.037	$0.037
Pebble consumption @ 5 lb. per ton ground	0.092	0.092	0.092
Liners consumption @ 1.2 lb. per ton ground	0.041	0.041	0.041
Power (steam) @ $168 per year	0.180	0.148	0.118
Power (electric) @ $62.50 per year	(0.067)	(0.055)	(0.044)
Repairs	0.024	0.024	0.024
Belting (2 years)	0.003	0.003	0.003
Labor @ $3.50 per day	0.007	0.007	0.007
Supplies	0.015	0.015	0.015
Total cost per ton ground (steam)	$0.398	$0.367	$0.337
Total cost per ton ground (elect. pow.)	$0.285	$0.274	$0.263

NOTE: Belts; 140 ft. of 24-in. belt per mill.
Labor: per day, 2 peones; 1 mechanic; ¼ mech. supt.; supt. and shift-bosses, proportional part.
Liners @ $0.034 per pound.
Pebbles @ $0.0184 per pound.

The costs given in Table VII. represent the average costs of grinding and regrinding. It was not possible to determine separately the exact cost of grinding the coarse sand from the batteries and the fine sand from the return cone, owing to the frequent necessity in the plant of switching one flow or the other from one mill to another. We may observe, however, that the cost of grinding fine sand is far greater than the cost of grinding coarse sand, owing to the reduced efficiency of the mills, and the greater wear of pebbles and liners, in the former case.

VII. SLIME-TREATMENT.

The slime-pulp, overflowing from the hydraulic classifiers, is composed of about 1 part of slime to 12 parts of mill-water. The pulp flows directly to the steel treatment-tanks, 12 ft. in height and 34 ft. in diameter, entering at the center through a wooden box, so arranged as to break the force of the flow. Caustic lime is added to the mill-pulp, just before it enters the

tube-mills, in the proportion of 12 lb. of lime per ton of ore. The slime settles completely, and the clear mill-water overflows and is pumped back to the mill storage-tanks.

Each slime-charge consists of from 62 to 85 tons (average 76 tons) of dry slime; and it takes about 12 hr. to collect the charge when 100 stamps are crushing through 30-mesh, and when one tube-mill alone is used for regrinding the sand; while it takes about 9 hr. when 40-mesh battery-screens are used, and when two tube-mills grind and regrind the coarse sand; and it takes only 5.5 hr. when crushing through 25-mesh, and keeping three tube-mills at work.

When a charge is completed, the slime is allowed to settle for 8 hr., at the end of which a 76-ton charge will have caked down to a depth of 50 inches. The pulp has an average specific gravity of 1.39, and will contain about 53 per cent. of water.

The clear mill-water is decanted as the settling takes place, thus avoiding any delay when the settling is considered to be complete. Lately, a larger percentage of very fine sand has been sent over with the slime, causing a far greater rate of settling of the slime.

The pulp is then set in agitation by a mechanical agitator, consisting of a horizontal wooden cross revolving in the center of the tank, 14 in. from the bottom, and at a speed of 7 rev. per minute. Two arms of this cross are 15 ft. long, and two are 7 ft. long. When the whole pulp is once set in rotation it takes about 6 h.p. to keep up the movement.

Ten minutes after the agitation has begun, a valve at the bottom edge of the tank is opened, and the pulp put into circulation by a Gwynne centrifugal pump, which elevates the pulp and throws it back to the center of the tank. Compressed air is fed near to the inlet of the pump to aërate the pulp. These centrifugal pumps do excellent work; but the cost of operating them is high, as they consume 14 h.p. The pump has to run 14.25 hr. for the complete treatment of each charge of slime. Reckoning the h.p-year at $150, the total cost of the pump-agitation is $0.14 per ton of slime.

When the pulp has been thoroughly mixed, it is sampled and the specific gravity is determined, so as to calculate the weight of each charge. Cyanide solution is then added to the pulp, and the treatment of the slime carried on.

The treatment followed at El Oro until July, 1905, graphically shown in Fig. 8, consists of 4 washes; but it was somewhat modified after the results of the tests to be described later.

1. *Preliminary Test No.* 1.—The first test made was to follow closely one slime-charge throughout its treatment, taking slime- and solution-samples every few hours, and keeping track of all the data which could have a bearing on the process of extraction. The results thus obtained were plotted in Fig. 7.

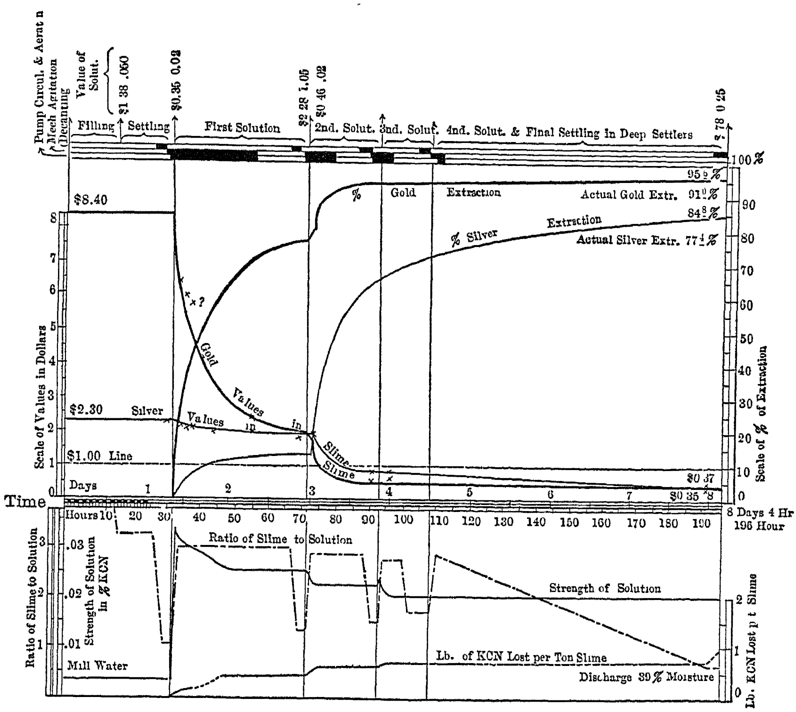
A few words on the methods followed in the graphic representation of an extraction-test will be found useful. For each test are given seven curves, representing the variations of the following data in function of the time:—

1. Undissolved gold-values contained in the slime.
2. Undissolved silver-values contained in the slime.
3. Percentage of absolute gold-extraction.
4. Percentage of absolute silver-extraction.
5. Ratio of weight of slime to weight of solution.
6. Strength of solution in per cent. of KCN.
7. Pounds of KCN consumed per ton of slime, since the beginning of the treatment.

The first 4 curves are grouped in one diagram, the 3 latter in a second one, placed below the first. Both diagrams have, as common abscissæ, the time elapsed since the first moment the charge began to settle and collect in the treatment-tank. The time (as abscissæ) is given in two units, days and consecutive hours; in the scale of the original blue prints, it was 15 hr. to the inch. The units of the ordinates, following the curve they refer to, are given by the scales on the sides of the diagrams. On the upper part of the diagram, heavy bars mark the periods of time during which the pulp was kept in agitation by means of the mechanical agitator, or by means of the Gwynne pump.

In Fig. 7 we can closely follow the process of extraction as it was practiced in the new plant up to July, 1905.

During the first 15 hr. the charge is collecting and settling; the settling continues until the 29th hour when, the mill-water having been decanted, the mechanical agitation is started, and soon followed by the pump-agitation. Up to the 31st hour the gold- and silver-values in the slime are considered to have remained constant. At the 31st hour 183 tons of 0.034-per cent.



Assay-returns marked by x.

Summary.

Quantity of slime treated,		94.4 tons.
Original assay-value,		Au. \$8.40 Ag. \$2.30
Absolute gold-extraction,		95.6 per cent.
Absolute silver-extraction,		84.8 per cent.
Actual gold-extraction,		91.0 per cent.
Actual silver-extraction,		77.4 per cent.
Absolute total extraction,		93.3 per cent.
Actual total extraction,		88.0 per cent.
Assay-values of tails,	Au. dissolved,	\$0.39
	Au. undissolved,	\$0.37
	Ag. dissolved,	\$0.17
	Ag. undissolved,	\$0.35
KCN consumption, per ton of slime,		0.96 lb.
Chemicals: lead acetate per ton of slime,		0.443 lb.
Quantity of solution used,		586 tons.
Quantity of solution precipitated,		403 tons.
Total time of treatment in slime-tanks,		80 hours.

Notes

Gold at \$20.67. Silver at \$0.50.

FIG. 7.—EXTRACTION TEST NO. 11, ON SLIME AT MILL NO. 2. (June 6 to 14, 1905.)

KCN solution were added, and the double agitation kept up until the 56th hour. As soon as the cyanide comes in contact with the slime, the gold begins to pass rapidly into solution, while the silver appears to be but slightly affected. The rate of dissolution of the gold, while great at the beginning, soon slacks off; and, if the first wash should be carried on indefinitely, the curve of undissolved gold-values would tend to become a horizontal line somewhere after the 100th hour. The silver-values, on the contrary, decrease in an almost linear way, as is also verified in the later experiments. It may be noticed that the gold- and silver-extraction proceeds continuously, even during the periods of settling.

From the 66th to the 69th hour, the first solution is decanted, and at the 71st hour a second addition of 0.034-per cent. KCN solution is added and mixed to the slime-pulp. During the first two hours, the gold is shown to be again rapidly dissolving, while the silver is practically unaffected. At the 73d hour lead acetate was added to the charge in the proportion of 0.44 lb. of lead acetate per ton of dry slime. The lead acetate was added in form of crystals, and hung in a sack of cocoa matting at the center of the tank, where it was struck by the flow of the pulp discharged by the circulating pump. Its effect on the gold- and silver-extraction is quite remarkable. The gold-values rapidly fall to 60c., coming near to their lowest possible limit, which is a function of the average diameter of the grain, and which limit for a 95-per-cent.-through-200-mesh slime is about 55c. on an original assay-value of $9 for the pulp.

The silver-values have also a rapid but somewhat less accentuated drop, and continue to decrease throughout the whole treatment. Even after 196 hours' treatment there will be some silver still passing into solution; and, at the moment of the discharge, about 30 per cent. of the undissolved silver in the tails could still be dissolved. This, however, could not be done with profit.

The attempt has been made to treat both sand- and slime-tailings with nitric acid, and then with *aqua regia;* and these tests have confirmed the results obtained by Mr. S. H. Pearce,[2]

[2] S. H. Pearce, *Transactions of the Institution of Mining and Metallurgy*, October 20, 1904, p. 46.

namely, that only a few cents in gold and about 0.2 oz. of silver per ton can still be brought into solution by such treatment. From the diagrams it is easy to see that, after the second wash, the gold-extraction comes practically to a stand-still, while the silver-extraction proceeds, though at a decreasing rate.

After the second solution had been decanted, two other washes were given to the slime; and with the fourth wash the slime was pumped over to the deep settlers, which are iron tanks, 20 ft. deep and 34 ft. in diameter, where, gradually settling, it caked down to 39 per cent. of moisture. Then it was discharged into the creek.

In practice, however, it was found almost impossible to give a fourth wash to the slime, because the plant was overcrowded. Hence, it was necessary to pump over with the slime the third solution, thus discharging into the creek the completely treated, but insufficiently washed, slime.

Concerning the consumption of KCN, we notice from the diagram that it is very great at the beginning of each wash, by reason of the dissolution of the gold and silver and the complex action of the cyanicides. All the experiments show that there is some loss in cyanide every time a solution comes in contact with the ore, even when all the values have already been brought into solution; and the loss in cyanide is proportional to the strength of the solution used. Therefore, the values ought to be dissolved in the shortest time possible, so as to reduce the losses in cyanide to the first few strong solutions used.

When the dissolution is economically complete, the succeeding washes will be carried on with the sole purpose of washing out the values. The wash-water is certain to become enriched with cyanide (which, by the way, is necessary for the precipitation in the zinc-boxes); but the loss of cyanide in the wash-water will be small, owing to the weakness of the solution. It is obvious that the cyanide-loss by discharging the tailings will be similarly reduced.

2. *Preliminary Test No. 2.*—A second test was carried on contemporaneously with the one just described, and the results were almost identical. In this test, however, the first solution used was stronger and the gold-extraction more rapid. The pumping and aëration during the first wash were suspended for

12 hr. after the charge and solution had become thoroughly mixed, and then started up again for 13 hours. The renewed aëration seems to mark an acceleration in the gold-extraction, though the drop of the tail-values is very close to the limit of error in sampling and assaying. Aëration is indispensable for rapid extraction; but this fact does not involve, as a consequence, the necessity of aërating the pulp during the whole period of agitation. The solution will rapidly absorb the maximum of oxygen soluble in it, and after one hour's pumping it will retain sufficient air-bubbles in emulsion to act as an oxygen-storage for a certain length of time.

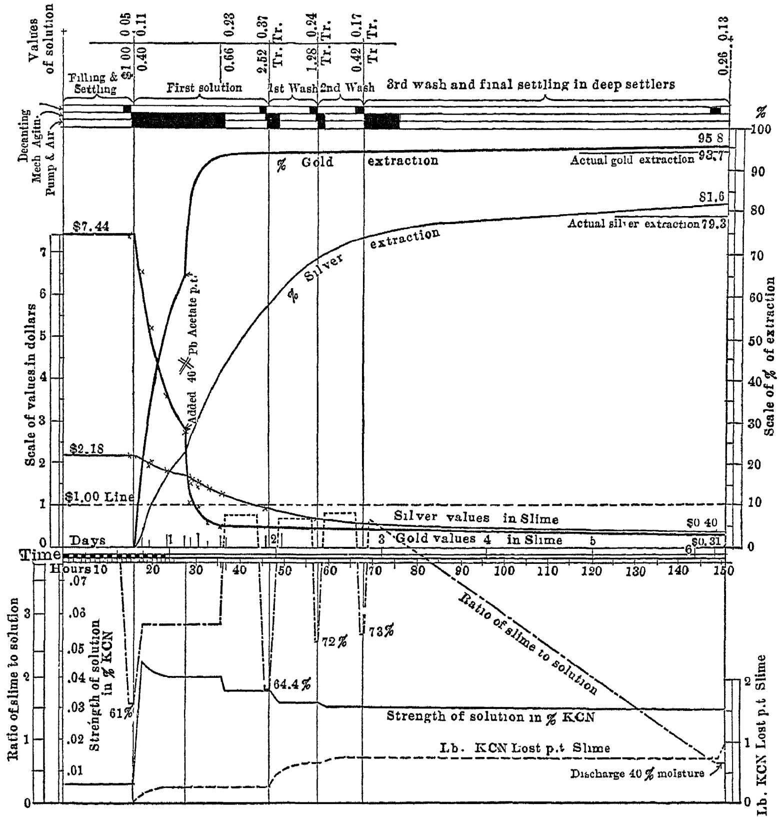
3. *First Modification of Slime-Treatment.*—The tests described in the preceding pages have shown us the marked action of the lead acetate on the rate of dissolution of the gold- and silver-values. Two further tests were made to determine whether it would be possible to complete the process of gold- and silver-solution during the first wash by adding chemicals at an earlier period of the treatment. The results of one of these tests are given graphically in Fig. 8.

In this test the lead acetate was added 11 hr. after the first cyanide solution had come in contact with the slime. During these 11 hr. the gold-extraction proceeds in a way similar to the one noticed in the test previously described. The silver-extraction is somewhat more rapid, a fact which may depend on some lead acetate being contained in the unprecipitated solution used as first wash. The addition of the lead acetate produces a sudden rapid extraction of the gold, and a far less accentuated one of the silver. In fact, the silver-values drop in an almost straight line throughout the whole treatment, independently of whether the slime is agitated or settling.

It is easy to see from the diagram that the first wash could just as well have been cut off at the 35th hour; and the 12 hr. included between the 35th hour and the end of the first wash could have been used for giving an extra wash to the charge.

This would have saved about 17c. per ton, by lowering the values in solution in the final wash. After the 58th hour the slime was twice washed, as rapidly as the settling of the slime would allow, and then it was pumped over to the deep settlers with its fourth wash-solution.

Contemporaneously with this test, a second test was made in



Assay-returns marked by x.

Summary.

Quantity of tons treated,	61.5 tons.
Original assay-value,	Au. $7.44 Ag. $2.18
Absolute gold-extraction,	95.8 per cent.
Absolute silver-extraction,	81.6 per cent.
Actual gold-extraction,	93.7 per cent.
Actual silver-extraction,	79.3 per cent.
Absolute total extraction,	92.6 per cent.
Actual total extraction,	90.4 per cent.
Assay-value of tails, Au. dissolved,	$0.16
Assay-value of tails, Au. undissolved,	$0.31
Assay-value of tails, Ag. dissolved,	$0.05
Assay-value of tails, Ag. undissolved,	$0.40
KCN consumption, per ton of slime,	0.96 lb.
Chemicals: lead acetate per ton of slime,	0.46 lb.
Quantity of solution used,	643.6 tons.
Quantity of solution precipitated,	457.0 tons.
Total time of treatment in slime-tanks,	60.0 hours.

Notes.

Gold at $20.67. Silver at $0.50.

FIG. 8.—EXTRACTION TEST NO. 14, ON SLIME AT MILL NO. 2.
(June 16 to 22, 1905.)

which mercuric chloride was used instead of lead acetate. In this case the agitation and aëration with the centrifugal pump was dispensed with as much as possible. After the charge had been thoroughly mixed for one hour, the agitation was carried on for 29 hr., merely keeping the pulp in rotation by means of the mechanical stirrers.

During the first 3.5 hr., after the solution came in contact with the slime, the dissolution of gold proceeded as rapidly as in the test previously described. The silver-extraction seems to be, as before, rather rapid. After 3.5 hours' treatment, mercuric chloride was added by suspending 16 lb. of $HgCl_2$ in a loosely-woven sack near the rim of the tank. The mercuric chloride, which was in broken pieces (some being as large as an egg), dissolved very slowly; after 10 hr. about one-third of it was still undissolved. The action of the mercuric chloride proved, however, to be exceedingly rapid, even with the small amount of it which could have dissolved during the first 3 hr. that it hung in the tanks. Ten hours after the first lot of $HgCl_2$ had been added, a further addition of 2 lb. of $HgCl_2$ was made; but this addition seems to have affected but slightly the extraction of silver.

This test shows that, during the 18 hr. following the 40th hour of treatment, the total useful work done was to extract only about 30c. in silver from the slimes. These 18 hr., therefore, were practically lost time; since the silver-extraction would have proceeded just the same if the slime had been settling; and they could have been used for giving an extra wash, which would have saved about 35c. per ton.

4. *Modified Slime-Treatment.*—The four tests described in the preceding pages give the evidence that, by using lead acetate or mercuric chloride at the beginning of the first wash, the extraction of the gold can be almost completed in 12 hr., and that 60 per cent. of the final extraction of the silver can be obtained during the same period of time.

The tests show also that the silver-extraction will continue during the following operation of washing-out the values by repeated settling and decanting, almost independently of the fact whether the slime is agitated or not.

It was, therefore, decided to modify the slime-treatment on slime-plant No 2. The outline of the treatment is as follows:—after decanting the mill-water from the already settled slime-

charge, the pulp is stirred up and mixed with about its equal weight of unprecipitated solution, so as to bring the total ratio of solution to slime to about 2.5 to 3. While the solution is flowing into the tank, the final strength of the solution is brought up to from 0.045 to 0.05 per cent. of KCN, by hanging the necessary amount of NaCN in a sack just under the flow of clear solution entering the tank. When there is an excess of strong solution in the sand-treatment plant, as after heavy rains, it is usual to pump some strong solution into the slime-tanks, so as to bring up the strength of the weak solution. Lead acetate (0.4 lb. per ton of slime) is dissolved at the same time and by the same method.

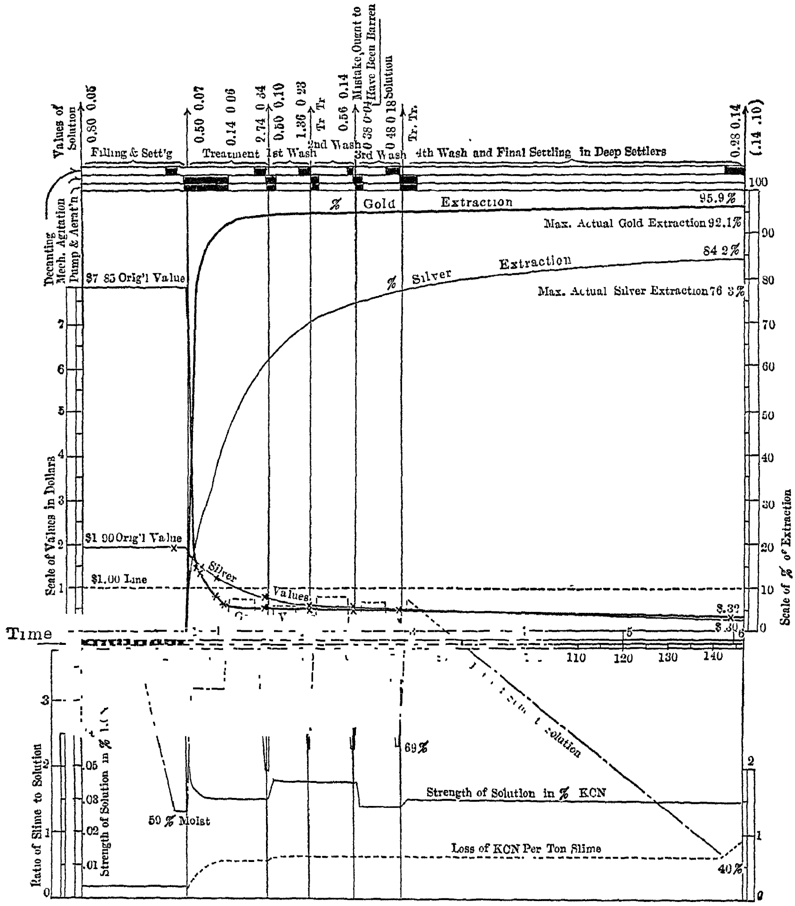
After the required volume of solution has been added to the charge, which takes about 1 hr., the double agitation by pump and stirrer is kept up for 7 hours. Then fresh, unprecipitated solution is added, so as to fill the tank completely. The agitation is then stopped, and the slime allowed to settle. After decanting the first solution as closely as possible, four additional washes are given to the charge as rapidly in succession as the settling of the slime will allow; and, with the last one, the pulp is pumped over to the deep settlers.

To diminish the amount of solution to be precipitated in the zinc-boxes, the first two solutions are unprecipitated solutions from the deep settlers. The value of such solution will average only 15c. in gold and a few cents in silver per ton, and therefore does not affect sensibly the final value of the last wash-solution.

It was customary at El Oro to keep the ratio of solution to slime as high as possible during the treatment. But the tests described in the preceding pages have proved that a rapid and good extraction can be obtained just as well when this ratio is as low as 2.5. The advantage of using a small bulk of solution during the first treatment is, that less cyanide and less lead acetate are needed to bring the solution to the desired degree of concentration.

Naturally, in washing-out the values, the ratio of solution to the slime has always to be kept as high as possible, to get a good extraction, because the capacity of the plant is not great enough to allow a great number of washes to be used.

In order to verify the efficiency of the modified treatment, a test was run on one of the slime-charges handled by the shift-bosses, and the results are given graphically in Fig. 9.



Assay-returns marked by x.

Summary.

Quantity of slime treated,	61.5 tons.
Original assay-value,	Au. \$7 85 Ag. \$1.90
Absolute gold-extraction,	95.9 per cent.
Absolute silver-extraction,	84.2 per cent.
Actual gold-extraction,	92.1 per cent.
Actual silver-extraction,	76.3 per cent.
Absolute total extraction,	93.6 per cent.
Actual total extraction,	98.2 per cent.
Assay-value of tails, Au. dissolved,	\$0.30
Assay-value of tails, Au. undissolved,	\$0.32
Assay-value of tails, Ag. dissolved,	\$0.15
Assay-value of tails, Ag. undissolved,	\$0.30
KCN consumption, per ton of slime,	0.90 lb.
Chemicals: lead acetate per ton of slime,	0.46 lb.
Quantity of solution used,	813 tons.
Quantity of solution precipitated,	459 tons.
Total time of treatment in slime-tank,	50 hours.

Notes.

Gold at \$20.67. Silver at \$0.50.

FIG. 9.—EXTRACTION TEST NO. 16, ON SLIME AT MILL NO. 2.
(June 26 to July 2, 1905.)

We shall only call attention to the fact that the 4th solution used was, by mistake, an unprecipitated solution, assaying 38c. in gold and 10c. in silver. If it had been a precipitated, and therefore practically barren, solution, the values of the final wash would have been 14c. in gold and 10c. in silver, instead of 28c. in gold and 14c. in silver per ton.

In Table VIII. are given some data regarding the slime-treatment, showing the increased saving obtained during the months of July and August, as compared with the months of May and June, recorded in Table VII.

TABLE VIII.—*Data of Slime-Treatment, Showing Increased Economy During July and August*, 1904.

	May.	June.	July.	August.
Average gold value of slime-tails...	$1.150	$1.130	$0.840	$0.570
Average silver value of slime-tails.	0.810	0.790	0.440	0.340
Average goldvalue of last wash...	?	0.527	0.180	0.143
Cost of pumping and agitating......	0.610	0.465	0.234	0.206
Total cost of treatment................	1.675	1.516	1.261	1.219
Total loss and expenses per ton of slime......	3.635	3.436	2.541	2.129

Preliminary tests have proved that lead chloride has the same effect on the rate of extraction as lead acetate. However, it has not been possible, up to this moment, to obtain quotations on the price of lead chloride sold in car-lots, nor has it been possible to try the chloride on a working-scale.

Table IX. gives, in a condensed form, the cost of treatment per ton of slime, as well as some data which may be found useful for reference.

TABLE IX.—*Cost, per ton, of Slime-Treatment at Mill No.* 2 (1905).

	July.	August.
Separators, ...	$0.0252	$0.0199
Sand-wheel, ...	0.0221	0.0129
Tube-mills, ...	0.1507	0.2796
Precipitation, ...	0.2128	0.1943
Cyanide, ...	0.2202	0.1637
Circulation-pumps, ...	0.1358	0.1398
Agitators, ...	0.0983	0.0658
Solution-pumps, ...	0.0626	0.0637
Retorting and melting, ...	0.0700	0.0514
Labor, ...	0.2221	0.1792
Other items, ...	0.0417	0.0484
Total, ...	$1.2615	$1.2186

Data of Slime-Treatment.

	Per Cent.
Moisture: Pulp flowing to treatment-tanks,	92
Charge collected (1st settlement),	53
Slime-tails discharged	40
Charging: Average charge per tank,	76.4 tons.
Time of collecting one charge:	
100 stamps, 40-mesh, 2.5 in. discharge, 1 tube-mill,	11 hr.
100 stamps, 30-mesh, 2.5 in. discharge, 2 tube-mills,	9 hr.
100 stamps, 30-mesh, 2.5 in. discharge, 3 tube-mills,	7 hr.
100 stamps, 25-mesh, 2.5 in. discharge, 3 tube-mills,	5.5 hr.
Sizing-Test: On 200-mesh (0.067 mm.),	from 5 to 12 per cent.
Through 200-mesh (0.067 mm.),	from 95 to 88 per cent.
Treatment-Tanks: Diameter,	34 ft.
Height,	10 ft.
Normal capacity,	9.079 cu. ft.
Thickness of plate,	$\frac{1}{4}$ and $\frac{3}{16}$ in.

	Hours
Time: Agitation of first solution,	8
First settling,	8
Total time for each wash,	9.5
Total time for collecting, treating, washing, decanting, losses of time, shut-downs for each 76-ton charge (average, July, 1905),	78
Time for complete extraction of gold,	12
Time of agitation by pumps and stirrers,	14.25
Time of settling in deep settlers,	from 3 to 4 days.

Solutions and Chemicals: Strength of solution,	0.05 per cent. of KCN.
Ratio of solution to slime (treatment),	2.5
Ratio of solution to slime (washing),	5.0
KCN consumption per ton of slime,	0.95 lb.
Lead acetate used per ton of slime,	0.45 lb.
Or mercuric bichloride per ton of slime,	0.20 lb.
Lime used per ton of ore crushed,	12.0 lb.
Assay-Values: Value of heads,	Au, \$7.00–9.00; Ag, 1.50–2.25
Value of tails (unwashed),	Au, 0.40–0.70; Ag, 0.25–0.45
Value of mill-water,	0.50–1.00
Value of first solution decanting,	2.40–3.00
Value of last wash,	0.15
Gold-extraction,	91 to 92 per cent.
Silver-extraction,	75 to 80 per cent.
Value of precipitated solution,	trace

Agitation: h.p. for running one centrifugal pump,		14.6 h.p.
Total cost per hr. one centrifugal pump,	\$0.76	
Cost of one centrifugal pump (San Francisco),	192.	
h.p. for running one mechanical agitator,		6 h.p.
Total cost per hr. for one mechanical agitator,	0.55	
Transmission-friction for agitation plant (about),		40 h.p.
Total cost per h.p-day,	0.46	

Labor, per day (sand and slime):
- 3 shift-bosses on 8-hr. shifts.
- 6 native tanqueros (tank-men) on 12-hr. shift.
- 2 pump-men.
- 2 separator-men.
- 2 tube-mill men.
- 2 lime-feeders.

VIII. Sand-Treatment—Mill No. 2.

The sand, separated by the two spitzkasten of the plant, is elevated by a sand-wheel, 41 ft. in diameter, and flows to the sand-receivers, which are nine steel tanks, 22 ft. in diameter and 10 ft. in depth.

The sand is distributed by Butters' distributors, and settles promptly, while the mill-water overflows through slats, carrying away about 65 per cent. of the slime contained in the pulp entering the sand-receiver.

Since July 21st, a 6-ft. cone has been placed at the discharge of the sand-wheel, and the coarse sand is separated and returned to the tube-mills, while the fine sand overflows to the sand-receivers.

The legend of Fig. 2 shows the diagram of the average sand collected under the present conditions and with the stamps crushing through 25-mesh. It is to such a class of sand that the tests hereafter described refer to.

The specific gravity of the sand is 2.55; when collected in the sand-receivers it will settle, leaving 40 per cent. of the total volume as interstitial spaces. When the interstitial spaces are completely filled with water, the water in the sand is 21 per cent.; if allowed to drain for 18 hr. the sand will still retain from 14 to 15 per cent. of water. The weight of the dry sand settled in the sand-receivers is 160 tons of dry sand for 8 ft. 6 in. actual filling.

When the sand has been drained for 24 hr., it receives three strong solutions, averaging from 0.20 to 0.30 per cent. of KCN. Then it is excavated by a Blaisdell excavator, and conveyed by Robins belt-conveyors to the steel treatment-tanks, which are 40 ft. in diameter and 78 in. in depth, 66 in. of which are available for filling. The sand is distributed in these tanks by a Blaisdell distributor. Each treatment-tank can be filled to the rim, when it will contain 282 tons of sand at 24.5 cu. ft. to the

ton. During the treatment, however, the sand will pack 10 per cent., and, at the end of the treatment, 22 cu. ft. of sand will weigh one ton.

The treatment consists in leaching the sand with a certain number of strong (0.3-per cent.) solutions, followed by a medium (0.15-per cent.) solution, and finally by a few weak (0.05-per cent.) solutions. When the treatment is completed, the tank is excavated and the sand conveyed to the dump. This is the outline of the treatment used up to August, 1905. To find out exactly what happens during the treatment, one charge of sand was followed closely throughout the whole process, and the results obtained are given graphically in Fig. 10.

The method of graphic representation used is similar to the one already explained for the slime. The scale of the ordinates (= time) had, however, to be reduced to 30 hr. per inch, in the original blue prints, which has been still further reduced in the illustrations given in this paper.

Another point in which the sand-diagram differs from the slime-diagram is, that each charge in the sand-treatment tank is composed of two sand-receiver charges. The first part of the diagram had, therefore, to be kept double, up to the moment in which the two sand-receiver charges were distributed in one sand-treatment tank. As this, however, brought a confusion of lines the data of per cent. extraction, KCN consumption, etc., were averaged, considering the two sand-receivers to be one unit-charge, not all parts of which had a synchronous treatment.

The curves, giving the gold- and silver-values of each separate sand-receiver charge, are, however, also given in dotted lines in the first part of the diagram.

1. *First Preliminary Test* (Fig. 10).—This test was run on a lot of 281 tons of sand, the composition of which was similar to the one given in Fig. 2, on page 89.

The charge was collected in two sand-receivers. The filling of receiver No. 1 took 20 hr.; the charge was allowed to drain 24 hr., then three washes of strong solution, amounting to a total of 0.48 tons per ton of sand, were percolated through the charge. The filling and treatment of the second sand-receiver, No. 3, followed by 20 hr. that of sand-receiver No. 1, and were kept as similar to it as possible. By reason of a shut-down,

both sand-receivers had to wait an average of 48 hr. after receiving the third wash before being excavated. At the 132d hour the excavation of sand-receiver No. 1 began, and, as soon as completed, was followed by that of No. 3. The amounts excavated from the two sand-receivers were, respectively, 159.8 and 121.5 tons. The average of the assay-value, percentage-extraction, etc., were figured out according to the relative tonnage of the two sand-receiver charges.

a. Gold- and Silver-Values.—The gold-values for both sand-receiver charges begin to decrease as soon as the first strong solution comes in contact with the sand, in a way similar to the one noticed for the slime, though the rate of extraction is very much smaller.

A sudden drop in values begins to take place only at the 132d hour, at the moment when the sand begins to be excavated, and therefore aërated.

From this moment on, the curve representing the gold-values assumes again a marked parabolic shape, the rate of extraction decreasing as the undissolved gold-values decrease.

The three main factors in the rate of extraction are the diameter of the undissolved gold grains, the strength of the solution, and the rate at which the solutions follow each other. It is very difficult to give to the assay-returns their true value, as the samples for assay cannot be relied on as closely as was possible with slime. The curve, therefore, had to be drawn following closely the assay-return.

We can notice, however, a certain drop of the gold-values at the 6th wash, probably caused by the fact that a strong solution was followed immediately by a weak solution. Any sudden change in the strength of the solution will produce an increased rate of gold-extraction. We see, from the diagram, that the gold-extraction was completed at the 250th hour, and that, from that moment, not one cent more was extracted during the 9 following days of treatment.

The silver behaves in a very different way from the gold. We can say, practically, that the values decrease proportionally to the time, and independently of the strength of the solution or any other factor; though actually, at the beginning, the rate of extraction was somewhat more rapid. The addition of lead acetate to the charge had a slight beneficial effect on the silver.

b. Values of Drained Solution.—The gold- and silver-values in the solution draining off from the sand are represented in the diagram by dotted curves.

Naturally, the increase in value of the solution is a result of the extraction from the sand. It is worth noticing that the maximum of the values of the draining solution always follows, in about 50 hr., the cause which produced an increased rate of extraction. In fact, we can notice on the diagram three maximums both for the gold- and for the silver-values of the solution. The first and most important maximum occurs 50 hr. after the excavation and aëration of the sand has taken place, the values of the solution rising up to a maximum of $14 gold and $2 silver. Then the values drop rapidly, to rise again slightly on the 12th day, or 50 hr. after two strong washes, alternated with a medium wash, have been given to the charge. Finally, during the 14th and 15th day we have again a rise in values, 50 hr. after another batch of strong solutions, following two weak solutions, was run on the sand.

The reason that the values of the solution follow the rate of extraction with a period of 50 hr. delay, is easily understood if we consider that about 40 or 50 tons of solution are kept back as moisture in a 300-ton charge of sand. Each time we add a wash of 15 tons of solution on the top of the sand-charge, we shall displace 15 tons of solution which were previously retained as moisture.

We can, therefore, imagine the 50 tons of moisture in the sand to be retained in four strata corresponding to four successive washes, and each time we add a new wash one stratum of moisture displaces the other.

c. Cyanide Solution.—The diagram shows that the strengths of solution used in the treatment were most irregular. This was done intentionally, in order to ascertain what bearing the strength of the solution, and the alternations of strong and weak solutions, had on the rate of extraction.

d. Conclusions.—The test just described leads to the following conclusions:—

(1) The rate of extraction is dependent upon the strength of the solution and the speed at which the washes follow each other.

(2) The extraction by strong solution is small for the tightly-packed sand in the sand-receivers.

(3) The aëration effected by excavating the sand has a very great influence on the rate of extraction, at least for a certain length of time.

(4) The gold-extraction will reach its maximum in a comparatively short time, and long before the whole of the soluble silver has been extracted.

(5) The silver-extraction will continue in an almost straight line, and apparently unaffected by the strength of solution and the rate of washing.

(6) Lead acetate, in small amounts, will not affect the gold-extraction, but will be beneficial to the silver-extraction.

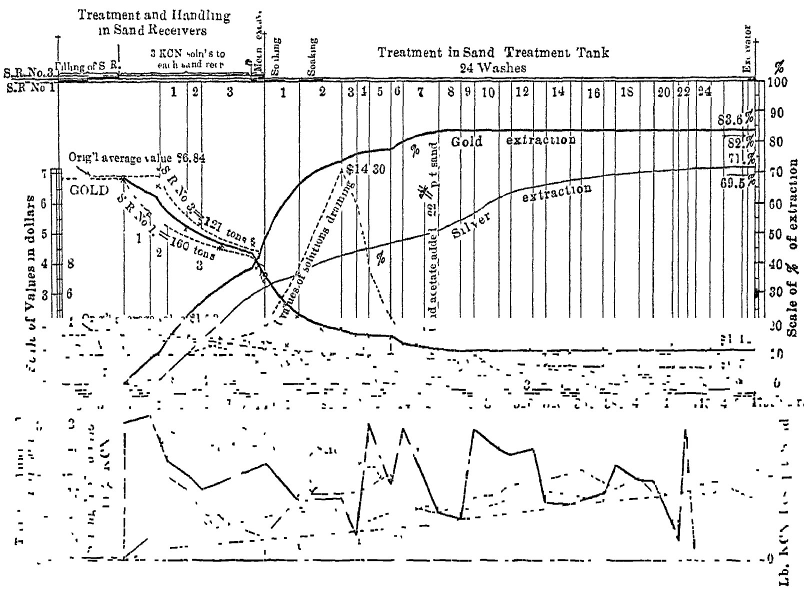
(7) Cyanide-consumption is rapid during the first 5 days (56.4 lb. of KCN per 24 hr.), after which the consumption becomes almost constant during the 13 following days (10.8 lb. of KCN per 24 hr.).

2. *Second Preliminary Test.*—The test previously described was followed by a second, the results of which are given graphically in Fig. 11. The main point aimed at in this test was to see whether it would be possible to increase the rate of extraction by excavating the sand as soon as it had received the first two preliminary strong solutions, and then giving to the sand a number of small washes in rapid succession.

From the foregoing it is evident, that, for the sand, the extraction depends chiefly upon fine grinding. The soluble gold is easily brought into solution, as is also the silver. The only improvement which can be made in the sand-treatment is to reduce expenses and losses, and to shorten the time of treatment required for the dissolution of the values, in order to increase the capacity of the plant or to get more time for dissolving the silver and for washing-out the dissolved values.

The diagrams on Fig. 11 can be easily understood after what has been said for test "T" (Fig. 10). The following facts, however, deserve attention :—

The undissolved gold-values show three marked drops: the first, after the 110th hour, when the first (3d) solution was run on the sand after it had been aërated during the transfer from the sand-receivers to the treatment-tank; a second drop takes place after the 131st hour, and a third one after the 163d hour, both due to the fact that a strong solution was followed by a



Assay-returns marked by x.

Summary.

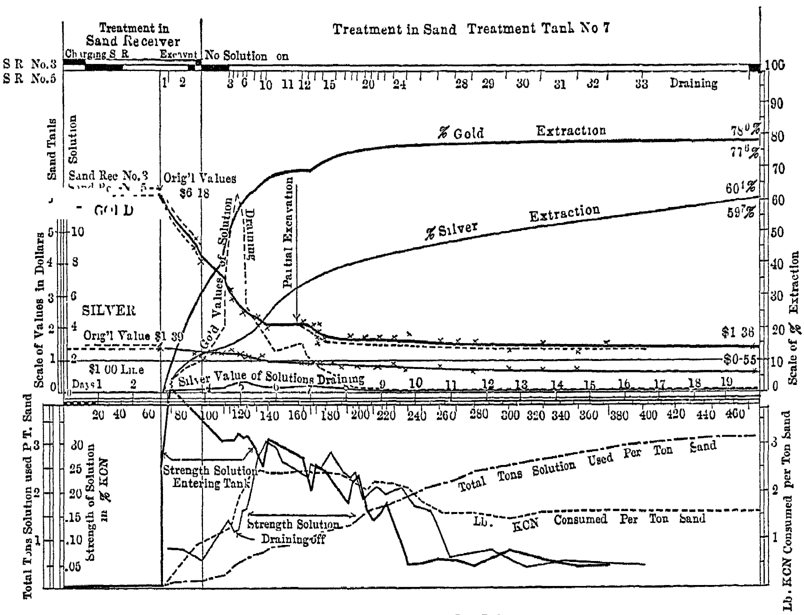
Quantity of sand treated,		281.5 tons.
Original assay-value,		Au. $6.84 Ag. $1.73
Absolute gold-extraction,		83.6 per cent.
Absolute silver-extraction,		71.0 per cent.
Actual gold-extraction,		82.9 per cent.
Actual silver-extraction,		69.5 per cent.
Absolute total extraction,		81.1 per cent.
Actual total extraction,		80.2 per cent.
Assay-value of tails,	Au. dissolved,	$0.05
	Au. undissolved,	$1.12[a]
	Ag. dissolved,	$0.02
	Ag. undissolved,	$0.50
KCN consumption, per ton of sand,		1.69 lb.
Chemicals: lead acetate per ton of sand,		0.22 lb.
Quantity of solution used per ton of sand,		1.74 tons.
Quantity of solution precipitated,	(1.74 × 281.5)	489.81 tons.
Total time of treatment,		475 hours.
Time taken for dissolution of gold,		196 hours.

[a] Average of 32 assays (4 A. T. each).

Notes.

Gold at $20.67. Silver at $0.50.

FIG. 10.—EXTRACTION TEST "T" ON SAND AT MILL NO. 2.
(July 1 to 20, 1905.)



Assay-returns marked by x.

Summary.

Quantity of sand treated,	236 tons.
Original assay-value,	Au. $6.18 Ag. $1.39
Absolute gold-extraction,	78.0 per cent.
Absolute silver-extraction,	60.4 per cent.
Actual gold-extraction,	77.6 per cent.
Actual silver-extraction,	59.7 per cent.
Absolute total extraction,	74.9 per cent.
Actual total extraction,	74.4 per cent.
Assay-value of tails, Au. dissolved,	$0.03
Assay-value of tails, Au. undissolved,	$1.35
Assay-value of tails, Ag. dissolved,	$0.01
Assay-value of tails, Ag. undissolved,	$0.55
KCN consumption, per ton of sand,	1.54 lb.
Quantity of solution used per ton of sand,	3.1 tons.
Quantity of solution precipitated, (236 × 3.1)	731.6 tons.
Total time of treatment,	478 hours.
Time taken for dissolution of gold,	120 hours.

Notes.

Gold at $20.67. Silver at $0.50.

FIG. 11.—EXTRACTION TEST "F" ON SAND AT MILL NO. 2.
(August, 1905.)

somewhat weaker one, and then again by a third solution of the same strength as the first.

In the 280 hr. following the 190th hr., not more than 14c. gold per ton of sand was dissolved, and the greatest part of it was dissolved in the first 72 hours.

The silver shows a fact noticed in several tests, namely, that the extraction begins to become more rapid as soon as the dissolution of the gold is almost complete. This may be due either to the fact that the cyanide will dissolve gold in preference to silver as long as there is gold still to be dissolved easily, or to the fact that some time must elapse before the reaction, causing the dissolution of the silver minerals, will begin.

At the 160th hr., one-eighth of the tank was excavated by hand to aërate one portion of the sand. The sand was thrown out on a platform and then returned to its place. Subsequent to this moment, samples were taken continuously from both the re-aërated part of the tank and from the untouched part. The assay-returns (see dotted line near the gold-curve) showed that the renewed aëration caused an increased rate of extraction; but such increase was small, and after a few days the gold-value of the sand in both parts of the tank had again become practically the same. This operation was repeated on a second tank with the same results.

3. *Modified Treatment.*—In consequence of the results given by the two tests described above, it was decided to modify the sand-treatment as follows:—

The sand collected in the sand-receivers was allowed to drain for 24 hr., when it was excavated and conveyed to the leaching-tanks without receiving any preliminary treatment.

The preliminary treatment was first adopted with the idea that it would be useful to have the sand-grains moistened with strong cyanide solution at the moment when the aëration by excavating takes place. Experiments, however, have proved that the extraction is just as rapid if the sand is treated by cyanide only after the aëration has taken place.

On the other hand, the preliminary treatment presents two drawbacks. The first is, that it is impossible to prevent some of the strong cyanide solutions used as preliminary washes to pass into the mill-water (see page 86) for two reasons: (1) the

drainage-plugs and valves will leak, or may be left open by mistake; (2) after excavating, the 6-in. filter-bed of coarse sand, which is left to cover the cocoa-mat filter, will retain 2.8 tons of cyanide solution as moisture, which, as soon as the sand-receiver begins to receive a new charge, will be completely washed out into the mill-water.

The second drawback is, that, after giving the preliminary washes, it is necessary to lose 24 hr. in allowing the tanks to drain before starting to excavate them.

A second point in which the treatment has been modified is, that the first 16 washes given to the sand are alternating strong and medium washes, and are followed by as large a number of weak washes, at 6-hr. intervals, as the capacity of the plant may allow or as may be considered convenient.

By these means the time necessary for the almost complete dissolution of the soluble gold (to within 20c.) has been at present reduced to about 90 hr., while formerly it took about 200 hr. to obtain the same result.

Other tests, not related in this paper, also prove that cyanide solutions, containing from 0.15 to 0.20 per cent. of KCN, will give as good and as rapid an extraction as a 0.30 per cent. of KCN solution. Therefore, the strength of the strong solution has been reduced to 0.25 per cent. of KCN, with the result that the cyanide consumption has been diminished.

We may say at this point, that, lately, a large amount of fine sand has been sent over with the slime to the slime-treatment tanks. On several occasions as much as 4 per cent. on 150-mesh and 9 per cent. on 200-mesh sand was mixed with the slime. On separating and assaying these sands, after treatment by agitation with 0.05-per cent. KCN solution, it was found that the extraction from the sands had been just as good as if they had been leached with 0.30-per cent. KCN solution in the leaching-tanks.

Extraction-tests, made on the sand treated by the modified treatment, show that the curve representing the gold-value of the sand is regular and continuous, and can be very closely represented by a parabola.

In the light of all that has been said in this paper on the problem of extracting the gold- and silver-values from the ore by cyanide, we see that the problem chiefly consists in the me-

chanical reduction of the ore to the finest possible state of division. There are no serious chemical problems to be resolved; and all the difficulties are on the mechanical side. The treatment of the slime, which, but a few years ago, seemed to present serious difficulties, is being gradually perfected; and the time will probably come when the sliming of the ore to the maximum point, economically practicable, will be the main object of a number of plants, as at present it is at El Oro.

The itemized cost and data of treating the sand at Mill No. 2, are given in Table X.

TABLE X.—*Cost, Per Ton, of Sand-Treatment at Mill No. 2.*

	July.	August.
Separators,	\$0.0227	\$0.0250
Elevators,	0.0400	0.0322
Tube-mills,[a]	0.2045	0.5286
Precipitation,	0.1918	0.2430
Conveyors,	0.0736	0.0561
Excavators,	0.0521	0.0855
Cyanide,	0.3593	0.2755
Solution-pumps,	0 0141	0.0159
Retorting and melting,	0.0631	0.0643
Other items,	0.1146	0.1635
Total,	\$1.1358	\$1.4866

[a] Tube-Mill No. 5 started August 1st, and the first cost of 14 tons of pebbles was charged to the month of August.

Data and Sand-Treatment.

Moisture of Sand: Well drained, ... from 14 to 15 per cent.
Charging: Charge collected in sand-receiver, ... 150 tons.
Charge treated in treatment-tanks, ... 285 tons.
Time for collecting 300 tons, ... 48 hr.
Sizing-Test: On 100-mesh, ... from 8 to 12 per cent.
Through 100- on 200-mesh, ... from 60 to 65 per cent.
Through 200-mesh, ... from 20 to 28 per cent.
Sand-Receivers (nine): 22 ft. in diameter by 10 ft. (0.25-in. bottom, $\frac{3}{16}$-in. body).
Maximum capacity, ... 160 tons.
Sand-Treatment Tanks (twelve): 40 ft. in diameter by 6.5 ft. ($\frac{1}{4}$-in. bottom, $\frac{3}{16}$-in. body).
Capacity, ... 285 tons.
Treatment: 16 alternated washes, medium and strong, ... 4 days.
16 weak solutions (more, if possible), ... 4 days.
Minimum time for treating, draining and discharging, . . . 9.5 days.
Average length of treatment, ... 18 days.
Time for complete gold dissolution (to within 20c.), . . . 90 hr.
Time for complete silver dissolution, ... 17 days.

Average rate of silver dissolution for a 300-ton charge per day,		$16.30
Average rate of silver after 10 days of treatment,		$6.60
Strength of solutions	Weak,	from 0.03 to 0.05 per cent. of KCN.
	Medium,	from 0.10 to 0.15 per cent. of KCN.
	Strong,	from 0.20 to 0.25 per cent. of KCN.
Total tons of solution per ton of sand,		from 2 to 3 tons.
KCN consumption per ton of sand,		1.5 lb.
Assay-Values: Values of heads,	gold,	from $6.00 to $8.00
	silver,	from 1.30 to 1.80
Value of tails,	gold,	from 1.10 to 1.80
	silver,	from 0.50 to 0.70
Probable economic limit to which the tails may be brought by fine grinding,	gold,	0.90
	silver,	0.40
Value of Solution:		
Precipitated strong solution,	gold,	from $0.03 to $0.15
Precipitated weak and medium solution,	gold,	trace.
Maximum of solution draining off,	gold,	from $9.00 to $14.00
	silver,	from 0.50 to 1.50
Last solution draining off,	gold,	from 0.10 to 0.35
	silver,	from 0.10 to 0.15
Excavators and Conveyors:		
Average rate of excavating sand-receivers by Blaisdell excavator, per hour,		from 90 to 100 tons.
Average rate of excavating per 40 ft. of treatment tanks,		from 90 to 100 tons.
Total cost of excavating (twice) per ton,		from $0.05 to $0.08
Total cost of conveying (twice) per ton,		from 0.05 to 0.07

IX. Precipitation and Precipitates.

1. *The Zinc-Room.*—The gold- and silver-bearing solutions, which are drained or decanted from the treatment-tanks, are collected in intermediate storage-tanks. The tanks receiving the solutions from the slime-treatment are provided with a filter-bottom to prevent the slime from passing into the zinc-boxes.

From the intermediate tanks the solutions flow to a zinc-room, 137 ft. by 73 ft., designed for precipitating the solutions from both the old and the new plant.

There are two rows of steel boxes, 30 of which are 16.5 ft. long, having five compartments; the available zinc space of each compartment is 4 ft. wide, 3 ft. long and 2 ft. deep; 14 other zinc-boxes, 16.5 ft. long, having 6 compartments 30 in. by 30 in. by 24 in. deep, were obtained from the 9 compartment boxes of the old precipitation plant. All the boxes are provided with 0.25-in. mesh screens set 4 in. from the bottom.

The boxes are set on a platform at 8 ft. from the floor; the precipitated solutions flow into central launders and pipes which lead to the sumps.

The boxes are equipped with bottom discharge-valves, which open into U-shaped steel launders, leading to a central launder, which empties into the 2 clean-up iron tanks, 6 ft. square and 4 ft. deep.

The precipitate is screened before entering the tanks, and from there it is pumped to a 24-in. Stillwell-Bierce filter-press, holding fifty 1-in. frames.

The acid treatment is done in two redwood tanks, 7 ft. in diameter and 4.5 ft. deep, lined with lead. The tanks are equipped with mechanical agitators, run by a 3.5-h.p. electric motor, which also runs a small blower for carrying-off the acid fumes through flues.

The wash-water and precipitate from the acid tanks are run into a cement sump, 7 ft. in diameter and 5 ft. deep, and a 4-in. by 6-in. bronze water-end duplex pump forces the precipitate into a 24-in. Johnson filter-press, holding 24 frames.

Four steam-jacketed drying-cars, set on masonry pillars, are used in case of accident to the filter-press, from which it might be necessary to take out soft cakes. One drying-car on wheels can be conveyed to any point of the room, and is run under the filter-presses when these are opened, so that the cakes fall directly into the car.

The entire floor of the zinc-room is cemented (6-in. masonry, 3 in. of concrete, and 1 in. of cement), and slopes from all sides to a cement sump at the center of the room.

On one side of the room are 6 melting-furnaces, built in one row. They are 26 in. in diameter and 4 ft. deep; the bottom for a distance of 6 in. tapers down to 19 inches. The furnace flues open into a common flue which leads to a large dust-chamber, 10 ft. by 5 ft. by 21 feet. The stack is 40 ft. high, and the draught is helped by a steam-jet from a 0.25-in. nozzle.

On the outside of the building is a track about 10 ft. above the level of the zinc-room floor, which offers the facilities of unloading the zinc shaving directly on to the floor where the precipitation-boxes are; this track is also used for emptying the sulphuric acid in the lead storage-tank, as well as for the coke in a coke-bin near the furnaces.

2. *Precipitation of Solutions.*—The solutions from the two plants are of equal strength, but the solutions from the new plant are lower in value, since a greater amount is required to wash out

the dissolved values from the very fine sand than from the comparatively coarse sand of the old plant.

There are 44 iron precipitation-boxes, of which 4 are used for strong solution (from 0.2 to 0.3 per cent. of KCN), 4 for medium solution (0.1 per cent. of KCN), and 36 boxes for weak solution (0.03 per cent. of KCN).

The average rate of flow through the boxes per cu. ft. of zinc shavings per 24 hr. is: strong solution, 1 ton; medium solution, 1.1 tons; and weak solution, 1.4 tons.

The precipitation of the weak solution is assisted by a "drip" of fresh 2.5-per cent. cyanide solution at the head of the box, added at the rate of 16 lb. of KCN per 24 hr., for the 36 weak-solution boxes.

Practically, all the precipitation takes place in the first 3 compartments, the last one being intended only to retain any precipitate which might flow off when the box is started up after a clean-up. The fresh zinc in this last compartment indicates whether the box is working properly or not.

Table XI. shows the manner in which the precipitation of the gold and silver takes place in the 5 compartments, as well as the variation of the strength of the solutions flowing through the compartments.

TABLE XI.—*Precipitation of Gold and Silver in the Zinc-Boxes.*

Precipitation and Cyanide Consumption in Zinc-Boxes.

	Head 1st Comp.		Head 2d Comp		Head 3d Comp.		Head 4th Comp.		Head 5th Comp.		Tail 5th Comp.	
	Au.	Ag.	Au.	Ag.	Au.	Ag.	Au.	Ag.	Au.	Ag.	Au.	Ag.
	$	Oz.	$	Oz	$	Oz.	$	Oz.	$	Oz.	$	Oz.
Values of sol a....	4.54	1.25	1.14	0.47	0.42	0.18	0.15	0.08	0.11	0.07	0.07	0.04
Cumul. % ext......	......	.....	75.0	62.4	90.8	85 6	96.7	93.6	97 6	94.4	98.4	96.8
Strength of sol	0.210 %		0.202 %		0.198 %		0.198 %		0.198 %		0.196 %	
Values of sol b	[illegible]	[illegible]	1.00	0.26	0.18	0.06	0.01	0.01	tr.	tr.	......	.....
Cumul. % ext......	......	. ....	66.6	58.1	94.0	90.3	99.6	98.4	100.0	100.0	100.0	100.0
Strength of sol...	0.128 %		0.124 %		0.122 %		0.116 %		0.112 %		0.106 %	
Values of sol c ..	0.62	0.33	0.36	0.18	0.25	0.17	0.11	0.04	tr.	tr.	......	. ...
Cumul. % ext......	.... .	......	41.8	45.4	59.6	48.4	82.3	87.9	100.0	100.0	100.0	100.0
Strength of sol...	0.054 %		0.052 %		0.050 %		0.048 %		0.046 %		0.046 %	

a Strong Solution. *b* Medium Solution. *c* Weak Solution.

3. *Clean-up.*—When the solutions are running regularly the strong boxes are cleaned every 4 or 5 days, the medium boxes about every 6 days and the weak boxes every 10 days. If, for any reason, any of the boxes are not working properly, the first two compartments receive a preliminary cleaning.

The general cleaning of the boxes is carried out as follows:—the flow is shut off and all of the zinc is taken out of the 1st, and piled up on the 3d, 4th and 5th compartments. The solution and precipitate are then drawn out of the 1st compartment and the screen cleaned and put back. All the zinc of the 1st compartment is then washed in the solution in the 2d compartment and replaced in the 1st. Great care is taken to stretch out the bundles of zinc-shavings and to pack them in the boxes in alternated layers running lengthwise and crosswise to the boxes. By this arrangement the solution, flowing through the zinc, finds everywhere an equal resistance, and has no tendency to channel. The bottom of the 1st compartment is packed with a 6-in. layer of good, firm zinc, on top of which are placed the zinc-shorts that have been returned from the clean-up screens; finally, the balance of the space is filled with fairly firm zinc.

The other compartments are treated in a similar way. The zinc-shavings are thus slowly moved from the last compartment towards the head of the box, and the fresh zinc is always added in the last compartment. Except on a monthly clean-up, the last compartment is left intact to act as a filter. No precipitate is carried over if the boxes are washed thoroughly and the flow through the boxes started slowly.

The precipitate is discharged from the bottom of the boxes into launders and collected in two clean-up tanks. Before entering the tank the precipitate is screened through a 50-mesh screen divided in three compartments. The heavy zinc-shorts settle in the center compartment, while the solution and the fine precipitate overflow on the sides and pass through the two other compartments. The precipitate settles rapidly in the tank, and the clear solution is pumped through the filter-press.

When all the boxes are cleaned and enough precipitate is collected in the tank to make a "cake," it is stirred up and the pump put to a moderate speed, which is not changed until the press is full with a hard cake. The pressure-gauge usually shows about 60 pounds. Steam is now gradually turned into the press until it begins to blow out of all the drainage-cocks. It takes from 0.5 to 1 hr. to fill the press, and 0.5 hr. to dry it out with steam. The press is allowed to cool for an hour, and then it is opened, the cakes dropping into the drying-car. If

the cakes still contain too much moisture for briquetting, they are further dried, then mixed with flux, briquetted and melted.

The zinc-shorts, which pass a 0.25-mesh screen and are retained on the 50-mesh screen at the clean-up tank, are sent to the acid tanks, where they are kept for the acid-treatment.

The sulphuric acid, diluted in the proportion of 1 of acid to 4 of water, is added to the zinc in small quantities at a time. The mechanical agitator keeps the zinc from settling.

After all the zinc has become dissolved, three or four washes of hot water are given to the residues, and the whole siphoned into a small cement sump or pumped direct to the filter-press, as previously explained, and handled in the same way as the screened precipitate.

Eight natives, working 9 hr., can clean up and wash all the zinc from 18 large and 3 small boxes. In one operation, 26,400 lb. of zinc-shavings are washed by 8 men; the precipitate, amounting to 1,760 lb., being handled mechanically from boxes to drying-car.

The monthly output consists of about 70 bars of bullion, each bar weighing 1,000 ounces.

4. *Screened and Acid Precipitates.*—The screened precipitate, after coming from the filter-press, contains about 15 per cent., and the acid precipitate about 25 per cent. of moisture. The precipitate is taken out of the press while still hot, allowed to dry during the night, and then mixed with 18 per cent. of ground borax-glass, 8 per cent. of sodium carbonate, from 6 to 8 per cent. of sand and 8 per cent. of lime.

The analyses of the screened precipitate and the acid precipitate are as follows:—

	Screened Precipitate.	Acid Precipitate.
Gold,	7.66	4.58
Silver,	47.70	23.99
Copper,	1.11	1.98
Lead,	......	4.00
Zinc,	12.99	7.79
Silica,	6.06	11.90
Iron, Aluminium,	3.03	2.12
Arsenic, Antimony,	none	0.41
Lime,	9.67	9.22
Sulphur,	......	9.94
Not determined,	11.78	24.07
Total,	100.00	100.00

On the average, the screened precipitate will yield 65 per cent. of bullion, while the acid precipitate will yield 32 per cent. of bullion, the lower percentage of the latter being largely due to the presence of calcium sulphate, which is hard to wash out entirely.

The copper, which is contained in the ore in very small quantities, gradually concentrates in the weak boxes. There is never enough of it in the solution to interfere with the precipitation of the gold and silver.

The bullion from the acid precipitate is always higher in fineness than that from the screened precipitate, because the gold, 61 per cent. of which precipitates in the first compartment, clings to the zinc-shorts more firmly than does the silver, which precipitates in a loose form and is chiefly collected with the screened precipitate.

During the last six months the average fineness of the bullion obtained from the screened and acid precipitates was:—

	Bullion from Screened Precipitates.	Bullion from Acid Precipitates.
	Fineness.	Fineness.
Gold	133.0	193.06
Silver	767.4	696.94
Total	900.4	890.00

The analysis of the bullion gave—Gold, 12; silver, 77.02; copper, 1.98; lead, 4.46; zinc, 32; silica, nil; iron and alumina, 0.35; total, 96.13 per cent.

5. *Briquetting Precipitates.*—The briquettes are made by a small upright machine run by compressed air. The cylinder is 14 in. and the briquette mold $2\frac{7}{8}$ in. in diameter. Under an air-pressure of 80 lb. the briquettes are compressed at a pressure of 2,000 lb. per sq. in.; but generally a pressure of 40 lb. of air is sufficient. In some cases, if a higher pressure be used, the briquettes will break in half when forced out of the mold.

The machine is fed automatically by a plunger-feed, which is at the bottom of a V-shaped hopper. The briquette is made by the down-stroke of the plunger, and is returned to the top of the mold by a return plunger, which is connected to the piston-rod by side rods.

Two men can handle the machine and make 10 briquettes a

minute. Much experimenting showed that the best binder was 8 per cent. of slacked lime, moistened by a saturated solution of borax-water, and added to the flux. The briquettes do not "dust," even if they get chipped in handling. If borax-water is used alone the precipitate must first be dried. Using lime, the precipitate should contain about 10 per cent. of moisture.

Crude petroleum answers fairly well as a binder, but it contains carbon, which, during the melting, reduces zinc and other metals, and produces a base bullion. Briquettes, 3 in. thick, average 2 lb. in weight.

6. *Melting.*—On the day of melting, the fires are started by the night-man towards 5 or 6 a.m., and as soon as the bottom of the crucible becomes dull red it is filled with the damp briquettes. In from 60 to 90 min. the briquettes in the bottom begin to melt, and, as the charge sinks in the crucible, fresh additions are made as long as there is no danger of getting the damp briquettes added directly into the molten mass, because, if this were done, the charge would spit and throw the slag and metal out of the crucible.

The crucibles are removed from the furnace by an air-lift. The tongs holding the crucible are suspended by a bail, hinged in such manner that the crucible, when two-thirds full of metal and slag, can easily be tilted.

It takes from 3.5 to 4 hr. to melt down and pour one charge. Five furnaces at one time are used for refining precipitate, and one for bullion; and one man and two stokers melt and refine 2,000 lb. of precipitate in 10 hours. The bullion and slag together are poured into conical molds, and, after solidification, the slag is broken off, and the bullion remelted in No. 100 and No. 200 crucibles, from which a dip-sample is taken. The bullion is poured into bars, each weighing about 1,000 ounces.

7. *Slags.*—The slag obtained from the smelting of the precipitate is sorted; the center part of the slag cones is remelted in a special cupola furnace, while the outer shell, as well as the slag from the top of the bullion bars, is remelted with from 5 to 10 per cent. of litharge-slag from the assay-office. The slag so remelted, assaying about $10 in gold, and from 15 to 20 oz. of silver per ton, is added to the charge that goes to the cupola furnace.

The ashes, sorted and screened to separate them from pieces

of coke and clinkers, are mixed with lime and water, made into cakes and allowed to dry. These ash-cakes are also melted in the cupola furnace.

The cupola furnace, which has been in operation only a few times, is proving to be a success; the blast used has been compressed air, but, as it was not easy to control the fire by such means, arrangements are being made to use a small blower.

The slag from the furnace assays from $3 to $5 in gold-value and 5 oz. of silver per ton. Data concerning the flue-dust are not, as yet, available. A sample swept from the floor of the dust-chamber assayed $210.90 in gold and 90 oz. of silver per ton, while a sample of flue-dust from the chimney of the old melting-furnaces assayed $372.06 in gold and 200 oz. of silver per ton.

A part-analysis of the slag from screened precipitates gave:—Gold, $24.38, and silver, 18.0 oz. per ton; copper, trace; lead, 1.05; zinc, 1.13; silica, 20.60; iron and alumina, 4.18; and lime, 7.60 per cent.

8. *Cutting the Zinc.*—The zinc-sheets, which are from 6 to 7 ft. long and from 3 to 4.5 ft. wide, are cut on a machine-lathe. They are wound on a wooden mandrel, about 14 in. in diameter and a little longer than the width of the sheets. The lathe is geared to 200 rev. per min., and the cutting-tool moves at a speed of 0.62 in. per minute. One man and a boy will cut from 1,000 to 1,500 lb. of zinc-shavings in 9 hours. The zinc-shavings are stretched out in beds, and then rolled up in shape of bales and kept in stock.

9. *Cost.*—Exact figures for the cost of precipitation in the new zinc-room are not yet available.

In the old mill, during the last six months, the average number of tons crushed per month was 10,380 tons. The cyanide-plant produced, per month, 29,859 oz. of bullion, averaging a total fineness of 889, or 4,030.35 oz. of fine gold, and 26,533.6 oz. of fine silver.

The cost of precipitating and melting, per ounce of bullion produced, and per ton of ore crushed, are:—

Precipitation.

	Cost per Oz. of Bullion.	Cost per Ton of Ore Crushed.
Labor,	$0.0094	$0.0272
Zinc,	0.0233	0.0767
Repairs, supt., and light,	0.0059	0.0169
Total,	$0.0386	$0.1208

Melting.

	Cost per Oz. of Bullion.	Cost per Ton of Ore Crushed.
Labor,	$0.0051	$0.0147
Chemicals,	0.0061	0.0175
Crucibles,	0.0034	0.0100
Fuel,	0.0027	0.0077
Repairs, supt., power and light,	0.0039	0.0113
Total,	$0.0212	$0.0612

	Per Oz. of Bullion.	Per Ton of Ore Crushed.
Zinc-consumption,	0.36 lb.	1.21 lb.
Sulphuric acid,	0.05 lb.	0.14 lb.

The Amalgamation of Gold-Ores.*

BY THOMAS T. READ, NEW YORK, N. Y.

(London Meeting, July, 1906.)

THE purpose of the following research, as originally planned, was to investigate the influence of temperature upon the plate-amalgamation process. In order to consider the amalgamation process intelligently, it was first necessary to learn the nature of an amalgam. In the performance of this task it was found necessary to consult a large volume of literature and to perform experimental investigations. The conceptions of the nature of amalgams thus obtained have so important a bearing on the amalgamation-process as a whole, as well as upon the possible influences of temperature, that it was advisable to include them in the treatment of the subject. The broader title is therefore used, even though some features of the amalgamation process have been treated very briefly and others entirely omitted, there being nothing new to present regarding them and their development not required for a clear presentation of the matter in hand.

I. HISTORICAL INTRODUCTION.

Gold was one of the earliest metals known to man. Occurring in the metallic state, it could be picked up in nuggets or washed, as gold dust, from the sand and gravel of streams. Interesting descriptions of the processes employed in early times for the recovery of gold may be found in Pliny's *Natural History* and in the treatise by Agricola on ores and minerals. But little progress in the treatment of the ores of gold was possible until efficient crushing machines were devised. This want was supplied by the introduction, about the beginning of the sixteenth century, of the stamp-mill to treat the ores of Saxony. Blanket-strakes, or similar devices, were used to catch the

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gold. Except in the gradual improvement of the machinery no important advances were made for many years. Hungary was the principal center of gold-milling in the eighteenth century. When the gold-fields of the southern United States were opened up, the processes used there were substantially the same as those employed in Hungary. But when the gold-fields of California were discovered, the great richness of the ores naturally stimulated inventive ability, and a legion of patent amalgamators resulted; these mostly survive at present only in the Patent Office records.

An improvement of lasting merit was the substitution of copper plates "amalgamated" or coated with a film of mercury, for the blanket-strakes, etc., until then employed to catch the gold liberated by crushing. This improvement is of such signal importance that it is to be regretted that the inventor is not known. The introduction of amalgamated plates into all the districts where gold-ores were being milled quickly followed, until at present their use is well-nigh universal for ores that are adapted to stamp-milling or dredging.

II. The Amalgams.

The amalgams have been a subject for research since the earliest days of the alchemists, and in few directions has so much work been done with such meager results in the acquiring of exact information. Research has proceeded along every practicable line; but it is not yet possible to say that the nature of amalgams is completely understood, although results obtained within the last few years have placed them in a much clearer light than ever before.

The assumption was early made that mercury forms an inter-metallic compound, or a series of compounds, with the metals with which it amalgamates. Many attempts have been made to isolate such compounds by the use of solvents, pressure, filtering, or volatilization, and many compounds have been described, although the most accurate work has shown in many cases that the results are not concordant enough to justify the assigning of definite formulæ. The heat of formation of amalgams also has been studied by several investigators, most notably Berthelot. It was found that in some cases heat was evolved and in others absorbed. A simple and complete ex-

planation of the results was not easy in most cases. The specific gravity and specific heat of amalgams have also received attention, the general conclusion being that amalgams are mixtures of a solid and a liquid. An investigation[1] of the thermal expansion has shown that where the percentage of the metal other than mercury is very small the amalgam acts like a solution, but with greater percentage of the other metal (tin, lead, or zinc) it is apparently a mixture of a liquid and a solid.

The study of the electrical properties of amalgams has been more thorough because of their use in the preparation of standard cells. Investigation of the electrical conductivity has, in general, gone to show that, for small percentages of foreign metals in the mercury, the effect is that of a solution; while for greater percentages, apparently a mixture of liquid and solid is present; thus confirming the results of the investigations of the thermal expansion and specific gravity. The study of the electro-motive force of amalgams has given more definite results. In the case of the alkali metals it has been demonstrated that more than one inter-metallic compound is formed with mercury. Haber[2] concluded generally that the metals in amalgams either were in the atomic state or formed compounds of the type Hg_mM.

The microscopic examination of polished surfaces of amalgams has so far yielded but unsatisfactory results, due to the difficulties arising from the presence of the liquid phase. The study of the freezing-point curves has been more productive, but is not entirely concordant with the foregoing work. In the case of the tin-amalgams, Van Heteren[3] has found that either tin, or a solid solution of mercury in tin, freezes out from the liquid down to $-34.5°$ C., at which point a transformation takes place, and the new form, "probably mixed crystals," freezes out to the solidification point of mercury. Roozeboom[4] has shown that, in the series Cd-Hg, above 77 per cent. of Cd mixed crystals separate out; below 67 per cent. of Cd other mixed crystals of the Hg type separate out; between these

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[1] Cattaneo, *Atti della Reale Accademia delle scienze di Torino*, vol. xxv., pp. 342-59, 492 (1890).

[2] *Zeitschrift für physikalische Chemie*, vol. xli., p. 399 (1902).

[3] *Zeitschrift für anorganische Chemie*, series 2, vol. xlii., p. 129 (1904).

[4] *Proceedings Koninklijke Akademie van Wetenschappen, Amsterdam*, vol. iv., p. 1 (1901).

ranges the action is more complex. Cd is insoluble in solid mercury.

The most complete study of amalgams by means of their cooling-curves has been done by Pushin,[5] who carefully traced the cooling-curves of the series, Hg-Bi; Hg-Cd; Hg-Zn; Hg-Pb and Hg-Sn. These curves showing no evidence of the separation of inter-metallic compounds, the author concluded that the series Hg-Zn and Hg-Bi are mechanical mixtures, and the series Hg-Cd, Hg-Sn and Hg-Pb are solid solutions. These results can scarcely be accepted without comment, but criticism does not find a proper place here. Mercury forms the eutectic for all the series, except Hg-Zn, where the eutectic occurs at about 2½ per cent. of Zn. Tamman[6] had already shown that for the series Hg-Zn and Hg-Bi the eutectic is very nearly pure mercury.

In conclusion, it may be said that with the alkali metals mercury forms a rather complicated series including several inter-metallic compounds.[7] With some of the other metals the amalgams consist of solid or liquid solutions or mixtures of both. Others are not yet completely understood.

III. Gold and Silver Amalgams.

1. *Native Amalgams.*—Naturally occurring amalgams of gold and silver have been found in some localities. Schneider[8] reports gold-amalgam occurring in small grains with the platinum of Colombia, and having approximately the composition Au_4Hg_5. An amalgam from the Mariposa region, California, analysed by Sonnenschein,[9] was nearly Au_3Hg_4 in composition. Silver-amalgam is more common, and numerous occurrences and analyses are cited in Dana's *System of Mineralogy.* The crystals of silver-amalgam are isometric, commonly dodecahedral, the analyses vary from Ag_2Hg_3 to nearly pure silver. Isometric crystals occurring in cavities filled with mercury, at

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[5] *Zurnal russkago physiko-chimiskago Obshestvo,* vol. xxxiv., p. 856 (1902). *Zeitschrift für anorganische Chemie,* xxvi., p. 201 (1904).

[6] *Zeitschrift für physikalische Chemie,* vol. iii., p. 441 (1889).

[7] Schuller, *Metallurgie,* vol. i., p. 433 (1904). Kurnakov, *Zurnal russkago physiko-chimiskago Obshestvo,* vol. xxxi., p. 927 (1899).

[8] *Journal für praktische Chemie,* vol. xliii., p. 317 (1848).

[9] *Zeitschrift der Deutschen geologischen Gesellschaft,* vol. vi., p. 243 (1854).

Sala, Sweden, analysed by Sjogren,[10] had the composition Ag_2Hg_3. But in the case of these amalgams it is only where there is excess of mercury present that we can be certain that no mercury has been lost by alteration. It is extremely difficult to free the amalgam from this excess of mercury; and the analyses do not seem to justify a more definite statement than that the composition of natural amalgams is somewhere near AuHg and AgHg.

2. *Artificial Amalgams.*—Joule[11] in 1850 described the preparation of a silver amalgam by the action of silver nitrate on mercury. He used a pressure of 72 tons (per square inch?) to press out the excess of mercury, and in this way produced a hard mass of shining crystals to which he gave the formula AgHg.

Henry[12] treated finely-divided gold with mercury, removed the excess of the latter by pressing through chamois leather, and treated the residue with hot nitric acid, thus obtaining yellow shining crystals, to which he assigned the formula Au_8Hg.

DeSousa[13] attempted to obtain amalgams free from excess of mercury, by holding them at a high temperature and ordinary pressure until there was no further loss of weight. The temperatures used were those of boiling mercury, sulphur and diphenylamine. At the temperature of boiling mercury, he obtained Au_9Hg and $Ag_{11}Hg$. In sulphur vapor Au_9Hg and $Ag_{13}Hg$ were produced, while with diphenylamine Au_8Hg and Ag_4Hg were found.

Merz and Weith[14] later repeated this work more carefully, conducting the heating in a stream of hydrogen or nitrogen. They found that the loss of mercury was apparently continuous, though very slow, in the later stages, and concluded that the amalgams are decomposed at the boiling-point of mercury and that the small amount of mercury retained is mechanically held.

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[10] *Geologiska Föreningens i Stockholm Förhandlinger*, vol. xxii., p. 187–190 (1900).

[11] *British Association Report*, 1850, p. 55.

[12] *Philosophical Magazine*, series 4, vol. ix., p. 468.

[13] *Berichte der Deutschen chemischen Gesellschaft*, vol. viii., p. 1616 and vol ix., p. 1050.

[14] *Berichte der Deutschen chemischen Gesellschaft*, vol. xiv., p. 1438.

Schnauss[15] and Dudley,[16] repeating the work in the same manner as De Sousa, obtained similar results.

Knaffl,[17] using HNO_3 of 1.35 sp. gr. at a temperature of 80° C., separated isometric crystals which he states to be nearly pure gold. Chester[18] found that, beginning with cold nitric acid of 1.2 sp. gr., gently heating, and finishing with hot acid of 1.4 sp. gr., elongated needles could be obtained which appeared to be hexagonal prisms terminated with pyramids and base. These contained about 6 per cent. of mercury. Both Knaffl and Chester were investigating the crystallization of gold, and paid little attention to the mercury-content of the crystals. Wilm[19] by the use of nitric acid obtained needles and prisms which contained from 9.87 to 11.45 per cent. of mercury. Fedorow[20] states that the crystals left by the treatment of gold-amalgam with nitric acid are prismatic, belonging to the hex-octahedral class, being elongated in a direction perpendicular to the octahedron faces. Fremy[21] states that a white crystalline mass having the composition $AuHg_4$ can be separated from gold-amalgam. Many other statements in regard to the composition of gold-amalgams can be found scattered through technical journals and text books, but these are usually vague or are based on very slender evidence.

Kasantseff[22] studied the action of nitric acid on solid and liquid gold-amalgams, concluding that the resulting alloys were not homogeneous in character. He also investigated the solubility of gold in mercury, by filtering a liquid amalgam through capillary tubes, and determined a solubility of 0.011 per cent. of gold at 0° C., 0.126 per cent. at 20° C. and 0.65 per cent. at 100° C. Dudley[23] checked this work by allowing mercury containing 0.1 per cent. of gold to stand quietly for several

15 *Archiv der Pharmazie*, series 3, vol. vi., p. 411.

16 *Proceedings American Association for the Advancement of Science*, vol. xxxviii., p. 145 (1889).

17 *Dingler's Polytechnische Journal*, vol. clxviii., p. 282 (1863).

18 *American Journal of Science*, series 3, vol. xvi., p. 29 (1878).

19 *Zeitschrift für anorganische Chemie*, vol. iv., p. 325 (1893).

20 *Verhandlung der russischen Kaiserlichen chemischen Gesellschaft*, vol. xxx., p. 455 (1893).

21 *Encyclopēdie Chimique*, vol. iii., p. 99.

22 *Bulletin de la Sociētē Chemique*, series 2, vol. xxx., p. 20.

23 *Proceedings of the American Association for the Advancement of Science*, vol. xxxviii., p. 160 (1889).

months at 20° C. in a tall glass cylinder. The mercury decanted from the top at the end of this time contained 0.068 per cent. of gold, and after filtering through boxwood contained 0.06 per cent. of gold, indicating that the higher figures are due to the presence of fragments of gold or gold-amalgam so minute as to pass through the filtering device employed.

Berthelot[24] studied the heat of formation of various silver amalgams; the results are significant, but are not easily explained. Ogg[25] has investigated the chemical equilibrium between silver nitrate solution and mercury. He found that beyond a certain concentration of silver in the mercury a solid phase separates out. By the use of methods analgous to those generally adopted for hydrated salts the composition was determined to be Ag_3Hg_4. This agrees very closely with the naturally occurring silver amalgam.

3. *Research Work.*—The work of Pushin on the cooling-curves of amalgams resulted in the acquiring of definite data in regard to their constitution. The attempt was therefore made to apply this method of study to the gold-amalgams. A nickel-copper thermo-electric couple was prepared and calibrated for the purpose; nickel-copper being employed rather than platin-rhodium because of its much greater sensitiveness, especially for the ranges of temperature within which it was intended to be employed.

The mercury used was chemically pure, having been purified by redistillation and treatment with acid. The gold employed was inquarted and parted with nitric acid, then dissolved in aqua regia, and precipitated by warming with ethyl alcohol, yielding a bright crystalline powder which was readily taken up by the mercury.

Since mercury forms the eutectic of the gold-mercury series, it is evident that, for mixtures high in mercury, the freezing-point curve coincides with the solubility curve of gold in mercury. The solubility of gold in mercury being known from the researches of Kasantseff and Dudley, it was easy to prepare an alloy of such composition that on cooling from the molten state the solid phase should begin to separate out near 100° C. This was placed in a small hard glass tube

[24] *Comptes rendus*, vol. cxxxii., p. 241 (1901).

[25] *Zeitschrift für physikalische Chemie*, vol. xxvii., p. 285 (1898).

surrounded by a mixture of magnesia and asbestos, the whole being enclosed in a crucible and carefully covered to exclude drafts. It was raised to a temperature of 150° C. and slowly cooled. The cooling-curve was perfectly smooth, without any breaks whatever. The experiment was repeated several times with always the same result. Mixtures containing more gold were then employed and the initial heating was carried to higher temperatures, finally using a mixture in which it was evident that there was a solid phase still present near the boiling point of mercury. So much mercury is lost by volatilization at these high temperatures that any results which might have been obtained would have been untrustworthy.

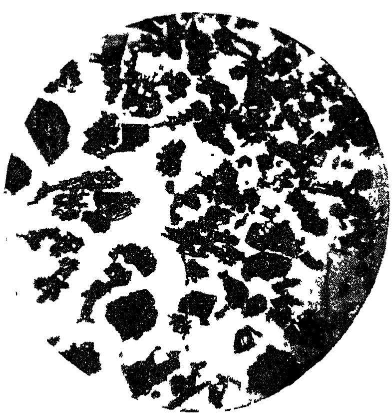
No breaks in the cooling-curve could be detected, and it became evident that this method of research was unproductive of results. Probably this was because the lag of the solid phase was not marked enough to produce a distinct jog in the curve. It was not possible to work with mixtures high in gold, for the mercury is lost by volatilization below their fusing-points at ordinary pressure, and the facilities were not at hand for working under high pressures. The study of cooling-curves, therefore, had to be abandoned, without having secured any definite results.

But this work had shown that a solid phase is present at ordinary temperatures with alloys low in gold. The constitution of this solid phase was the next subject of investigation. It was inferred that if an alloy was prepared in which the gold was entirely in solution in the mercury, and this alloy subjected to the action of a solvent which would dissolve the mercury, but not attack the solid phase, then we would get practically the effect of the freezing-out of the solid phase by the removal of the mercury with which it was in equilibrium, finally leaving only the solid phase present.

Nitric acid was judged to be the most likely solvent to yield good results. An excess of the finely-divided gold was added to mercury, and the mixture held at 100° C. The resulting liquid was filtered off and treated with nitric acid of different strengths, both at 100° C. and at ordinary temperature. The results were very variable. Acid of 1.42 sp. gr. at 100° C. gave a brownish powder, of indistinct structure, which was nearly pure gold, becoming bright on ignition. With acid of

less and less strength the solid contained more mercury, and with 1.1 acid the residue contained 13.62 atomic per cent. of mercury (*i. e.*, 13.62 atoms of mercury in 100 atoms of the alloy). Numerous determinations on this residue showed it to be fairly uniform in composition throughout.

The residues were examined under the microscope by reflected light, using magnifications of 40 and 60 diameters. The powders from the stronger acid were indistinctly crystalline, apparently isometric where the form could be distinguished. The residue from the 1.1 acid was made up of distinct isometric crystals of a golden color, the octahedron and cubo-octahedron being the forms observed. Scattered throughout these



Magnified 60 Diameters.

FIG. 1.—GOLD-MERCURY SERIES. ISOMETRIC CRYSTALS, SOLID SOLUTION.

were a few long needles of a bright brassy yellow; their form could not be determined beyond that they were apparently elongated prismatic crystals. Good photographs of this crystal mixture could not be obtained because of the nature of the material, but Fig. 1 shows its general isometric character.

With the use of nitric acid of 1.05 sp. gr. different results were obtained. The entire residue consisted of the brassy yellow crystals, and these were now so well developed that they could be distinctly seen to be hexagonal prisms terminated with pyramids and the basal pinacoid. Their surfaces were bright, and they had evidently not been attacked by the acid. On

analysis they proved to contain 17.44 atomic per cent. of mercury. They are shown in Fig. 2.

Subsequently to reaching these results it was discovered that A. H. Chester[26] had described hexagonal crystals of gold-amalgam, stating that they contained about 6 per cent. of mercury. Fortunately Chester's original material was still available for study, and upon examining it the pitted and corroded surface of the crystals was ample evidence that mercury must have been lost by the use of too strong acid. Chester used 1.2 acid at 18° C. at the start, finally finishing with strong hot acid, hence the composition obtained.

Magnified 40 Diameters.

FIG. 2.—GOLD-MERCURY SERIES. HEXAGONAL CRYSTALS, INTER-METALLIC COMPOUND OR SOLID SOLUTION.

An attempt was made to study polished surfaces of the gold-mercury alloys. The surfaces were obtained by casting on glass and mica. In every case the excess of mercury present coated the crystals, giving only indistinct, rounded outlines from which nothing could be inferred.

Having thus learned the degree to which mercury is soluble in gold, the solubility of gold in mercury was next investigated. The differing results of Kasantseff and Dudley have already been cited. The methods used in checking these results were two in number: adding an excess of gold, heating to a higher temperature and filtering at the temperature desired; and sus-

[26] *American Journal of Science*, series 3, vol. xvi., p. 29 (1878).

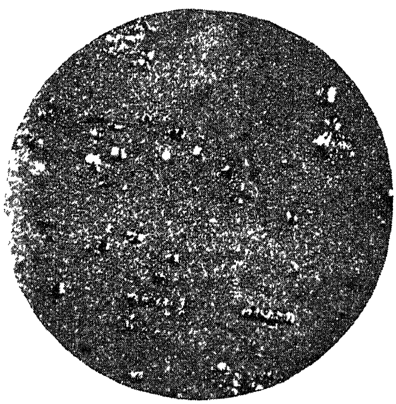
pending a thin sheet of gold below the surface of the mercury, holding the two for several days at the temperature at which it was desired to determine the solubility, and then analysing the mercury. In the former method fine cambric and close-grained chamois were used for the filters. It was found that the apparent solubility varied with the closeness of the filter, the pressure used in filtering, and also the degree to which the liquid had previously been heated, thus sustaining the contention of Dudley that the apparent solubility is due in part to the presence of minute solid particles which have passed through the filter. The results, though irregular, indicated that the solubility is at least not higher than the figures given by Dudley, namely, 0.06 per cent. at 20° C. This conclusion was confirmed by the second method, which also gave a solubility of 0.25 per per cent. at 100° C. The first method gave, for the solubility at 100°, 0.35 per cent. In determining the solubility by diffusion, the liquid was held a few days at 100° C.; but the diffusion may perhaps not have completed itself, and the true solubility may be the mean of the two figures.

The solubility of gold in mercury is very low indeed. Judging from the direction of the curve, it is zero at the freezing-point of mercury, so that pure mercury is the eutectic of the gold-mercury series.

The results, though not entirely satisfactory, seem to justify the conclusion that in the gold-mercury series, when the content of gold is high, a solid solution of mercury in gold, which is isomorphous with gold, separates out. This solution may contain as much as 13 atomic per cent. of mercury. In the lower ranges of the series, what seems to be an inter-metallic compound (holding gold or mercury in solid solution), containing 17.44 atomic per cent. of mercury, separates out: this crystallizes in the hexagonal system. From results on other amalgams, there remains a question as to whether this is really an intermetallic compound rather than a solid solution differing from the first. This question, and others not completely answered in the foregoing work, can be decided only after a large amount of very careful work. Their importance as bearing on the question actually under investigation did not appear to be great enough to justify undertaking such work at this time. Finally, gold is practically insoluble in solid mercury, and its

solubility in liquid mercury for ordinary ranges of temperature is very slight indeed.

The investigation of the silver-mercury series presents all the difficulties of the gold-mercury series, with the added one, that no solvent could be found which would attack mercury without attacking the solid phase. The only results worth mentioning were obtained by placing a solution of silver in mercury on silver plates and allowing the silver plate to absorb the excess of mercury. In this way the crystals were obtained that are shown in Figs. 3 and 4. These show the presence of two series of crystals, one apparently isometric and the other prismatic, with some modifying faces. From this evidence it may be inferred that silver is similar to gold in the forms which separate out from its series with mercury. The crystals were too small to analyse.



Magnified 40 Diameters.

FIG. 3.—SILVER-MERCURY SERIES. ISOMETRIC CRYSTALS.

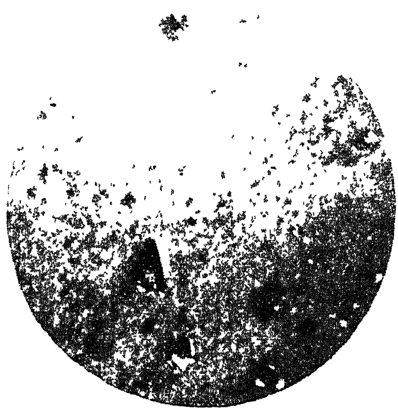
In this connection it must be remembered that Ogg[27] has shown that silver forms with mercury the inter-metallic compound, Ag_3Hg_4.

IV. MECHANICS OF THE AMALGAMATION PROCESS.

By the foregoing considerations the belief is confirmed that amalgamation is a physical rather than a chemical process. The solubility of gold in mercury is almost negligible, and the

[27] *Zeitschrift für physikalische Chemie*, vol. xxvii., p. 285 (1898).

diffusion of the mercury into the gold, forming a solid solution or inter-metallic compound, is a rather slow process, unless the gold is very finely divided, whereas the "catching" of the gold by the mercury is almost instantaneous. It is from a study of the physical features of the process that information is to be derived regarding the proper conduct of the operation. Yet it must not be forgotten that although amalgamation is, with respect to any given particle of gold, an almost instantaneous process, with respect to the total mass of ore it is a continuous process, and any change, however slow, must in time produce either beneficial or detrimental effects. All the forces which



Magnified 40 Diameters.

FIG. 4.—SILVER-MERCURY SERIES. PRISMATIC CRYSTALS.

operate must therefore be considered, and their relative effects determined.

With the exception of the tellurides, gold occurs in its ores as metallic or free gold, though frequently alloyed with greater or less amounts of silver, and, less frequently, with some other metals. Whether the tellurides of gold and silver are alloys or compounds is not a proper question for discussion here. The ores of gold are usually divided into two classes, "free milling" and "refractory" ores.

The closest practicable definition of these is that, in a "free-milling" ore, the gold occurs in such a form that it is readily and pretty completely recovered by crushing and amalgamation, while in refractory ores this is not the case. That the re-

fractoriness of an ore may arise from one or more of a variety of conditions, will be easily seen after outlining the physical features of the amalgamation of a "free-milling" ore.

The purpose of the crushing operations, usually performed in a stamp-mill with water, is to free the gold from the adhering gangue. The action of a stamp has been aptly likened to that of a hammer cracking a nut and liberating the enclosed kernel. If the stamp has done its work properly, the pulp, as it passes through the battery-screens, consists of particles of gangue and particles of free gold. This pulp is caused to pass over copper plates coated with mercury in a thin film. No mineral matter other than metal is susceptible of being wetted by the mercury, and, in addition, all the ore-material other than the metal is much lower in specific gravity than mercury, and hence flows over the mercury surface without being affected by it. The gold, on the other hand, on coming in contact with the mercury is immediately wetted by it, just as any ordinary solid is wetted by water; the action of the two liquids, water and mercury, being identical from a physical standpoint, and their apparent differences due to the differences in their physical constants. This action is assisted by the fact that gold has a greater specific gravity than mercury, and therefore readily sinks beneath its surface; but this feature is not so important as earlier writers[28] have claimed, for silver and the other lighter metals are readily amalgamated when their surfaces are in proper condition to be wetted by the mercury.

The gold particle, therefore, sinks beneath the surface of the mercury film which coats the plates, or, in mill-parlance, it is "caught." If the diameter of the particle is smaller than the thickness of the mercury film, it produces no disturbing effect on the latter. If, however, its diameter be greater, as will always be the case with coarse gold, the condition of affairs produced is as shown in Fig. 5.

The mercury rises over the grain of gold. But, as is well known, every liquid has a tendency, due to surface-tension, to provide itself with a horizontal upper bounding-surface. In the attempt to do this in that part of the surface which is ele-

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[28] Church, *Scientific American*, Oct. 7, 1871. Raymond, *Mineral Resources West of the Rocky Mountains*, 1872, p. 422.

vated by the gold grain lying beneath, the pull of the surface tension is resolved into components which press down the gold grain against the plate with a force which is considerable, compared with the dimensions and weights involved. This force begins to be exerted the instant the grain of gold is wetted by the mercury, and continues to operate until the grain is drawn beneath the surface; it constitutes the so-called "attraction" which mercury possesses for gold. Its exact amount may be computed, knowing the size and shape of the grain, the thickness of the film and the temperature. Under the influence of this force, especially if it is assisted by the splashing or dropping of the pulp, the grains of gold are caused to adhere to each other and to the plate with great tenacity.

A third factor, which is of great importance in causing amalgam to become hard, is a slow molecular flow, or some action of similar character. An amalgam of finely-divided gold and

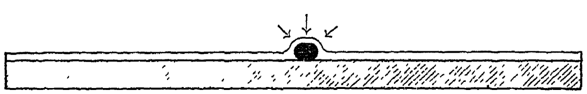

FIG. 5.—EFFECT OF CONTACT OF A GRAIN OF GOLD WITH A THIN LAYER OF MERCURY.

silver, made up to the consistency of a soft paste, will, after standing several days, become quite hard and rigid, but after kneading with the fingers assumes its original consistency. On standing again, it becomes as hard as before; and this operation may be repeated indefinitely. The hardening is much accelerated at higher temperatures, and, while it may be in part due to agglomeration caused by slight changes in the amount of mercury in solution in the silver or gold, resulting from the daily variations of temperature, yet chiefly it seems to be due to a movement of the particles of the metal, causing cohesion. Under the influence of these three factors the amalgam accumulated on a plate sometimes becomes quite hard, requiring considerable force to remove it.

It follows, accordingly, that any condition which tends to make the gold particles less easily wetted by the mercury has an important detrimental influence on amalgamation. Such conditions are many and varied. The influence of grease of any sort

is so well known that it needs only to be mentioned. If the gold has not been liberated by the crushing, it cannot, of course, be caught; and if it has been only partially liberated, the tendency of the mercury to hold the gold particle may be overbalanced by the tendency of the current to sweep along the adhering particle of gangue. Gold alloyed or combined with other substances is frequently rendered by them incapable of being wetted by the mercury. This is the case with the tellurides. Some writers have asserted that gold occurs in certain ores as a sulphide, but the evidence adduced to prove such a contention is inconclusive. Gold alloyed with bismuth is similarly affected, and there may, perhaps, be other cases. Much more common, especially in ores which have been subject to alteration, are adherent coatings of minerals which are not wetted by the mercury. These are commonly oxides, but may be sulphides or other gangue-minerals. One case has been noted where the gold particles were coated with a film of chalcedonic silica. Such coatings are often difficult to remove, and it may be necessary to resort to some process other than amalgamation. A more lengthy discussion of them and their effects can be found in the references given below.[29]

Another cause suggested for the refusal of mercury to wet gold at times is, that the latter may be in a strained state. Egleston[30] reported that gold, after hammering or rolling until it was brittle, would not amalgamate, though it had been cleaned with acid. T. K. Rose[31] reports that he was not able to confirm these observations. On rolling down pure gold until it was extremely brittle, I found that it would not amalgamate, even after cleaning with acid. But, on the surface being carefully cleaned with No. 0000 French emery-paper, it readily amalgamated, demonstrating that the refusal of the mercury to wet its surface was due to the dust and dirt which had been pressed into it, and not to the strained condition of the metal.

Very fine gold may be carried along so rapidly by the current as not to come into contact with the mercury. To prevent this as far as possible, the outside plates are set at such a grade that as little water may be used as is consistent with se-

[29] Egleston, *Metallurgy of Gold, Silver and Mercury*, pp. 584–95. McDermott and Duffield, *Losses in Gold Milling*.

[30] *Trans.*, ix., 648 (1881).

[31] Rose, *Metallurgy of Gold*, p. 149.

curing free movement of the pulp. Even if brought into contact with the mercury, such gold may refuse to become wetted, because the mercury is not able to displace the air or liquid film already adhering to its fine particles. The operation of similar action in the case of some sulphides in the various flotation processes is already known. The refusal of mercury to wet very fine gold has been noted by many writers, and Louis[32] has ascribed it to the existence of an allotropic modification of gold which is not wetted by mercury. This may be the case; but very finely-divided silver also refuses to be wetted by mercury, and it is perhaps better to leave the matter as an open question until further proof can be adduced.

It is usually the case that mercury is fed into the stamp-battery in order that, during the agitation of crushing, the gold may be brought into contact with the mercury, each grain being thereby thoroughly wetted. By the violent agitation during crushing, the mercury is subdivided into very small globules, thereby greatly increasing the chances that every gold particle will be brought into contact with the mercury and receive an adherent coating, which practically insures its being "caught" by the plates. This is for a triple reason: the weight of the particle is greatly increased; it is made more nearly spherical and correspondingly less likely to be carried off by the current of water; and, finally, the mercury film of the plates more readily "catches" a wetted particle than a dry one. To express the last statement in another way, mercury has less surface-tension against mercury than it has against gold. This subdivision of mercury into fine globules is one of the most important functions of the stamp-mill, and many authorities ascribe the excellence of stamp-milling as a process for the treatment of gold-ores principally to this feature of the operation.

But disadvantages are also introduced by this practice. The most important of these are the losses brought about by the "sickening" and "flouring" of the mercury. This arises chiefly from one or all of three causes. The first is purely mechanical. By the agitation in the battery the mercury is broken up into such small drops that the surface-tension of each is enormous, compared to its weight. This great surface-tension[33]

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[32] *Trans.*, xx., 182 (1891), and xxiv., 705 (1894).

[33] The surface-tension of mercury against water at 20° C. is 418 dynes per linear centimeter. Quincke, *Wiedermann's Annalen*, 1886, p. 219.

then keeps the globules perfectly spherical, and when they come into contact it is only at a point rather than along a line or upon a surface; the tendency of two globules to coalesce when they come into contact, or to sink beneath the mercury surface, is therefore very slight, and they are carried away with the pulp and lost. The tendency to settle is also very slight, just as a pebble, which falls to the earth rapidly, will, after being ground to dust, float in the air indefinitely.

The second cause is also mechanical, and arises from the presence of certain minerals which yield very fine slimes that adhere to the mercury surface, though not wetted by it. When covered with this fine coating, the globules can scarcely be made to coalesce with each other or with the mercury or the plates, and their loss is almost unavoidable, though the use of mercury-traps is of considerable assistance. An entirely similar action with water can be seen by sprinkling drops of water on a surface previously coated with lycopodium powder. Each drop becomes coated with the powder, taking on a spherical shape, if small, and moving about with the "quickness" of mercury. The drops can scarcely be made to coalesce, even if brought together with considerable force. The oxides of manganese and certain sulph-antimonides and sulph-arsenides are the worst offenders in this regard in amalgamation. It is reported that, in certain instances, the addition of various chemicals has been beneficial in preventing this action; but the data are not conclusive.

The third cause is chemical, and is a double source of loss. Soluble salts in the ore react with the mercury, causing it to be lost in solution, or precipitating other elements into it, causing it to become "sick." "Sickness" is of two kinds. Any other metal alloyed with the mercury makes it much more viscous and less "quick." A practical use is made of this fact in astronomical and physical work, where a mercury surface is used as a reflector. Mercury alone is too sensitive, vibrating constantly; but the addition of a small amount of tin causes it to stiffen enough to be perfectly satisfactory. In the case in hand this action is undesirable, for it causes the mercury to subdivide so readily that "flouring" takes place to an injurious degree. It is also undesirable for two other reasons: the fine slimes previously mentioned adhere much more readily to

the surface of "sick" mercury; and the metals in the mercury frequently reoxidize, forming a coating which almost prohibits the coalescing of the drops, or their being caught on the plates.

So far we have been concerned with actions that are comparatively rapid. But there are also others, of a slower rate, which remain to be considered.

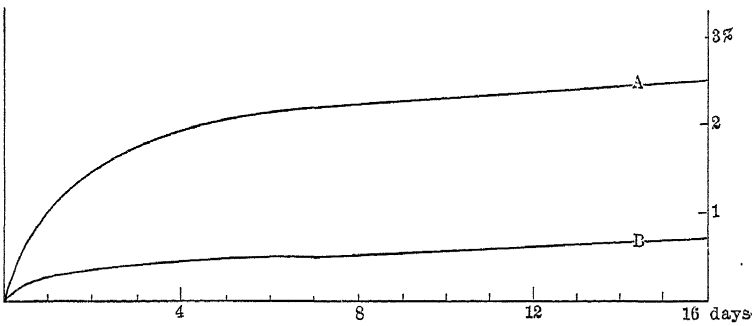
The first is the absorption of the mercury by the other metals taking part in the process. It has already been pointed out that mercury is soluble in gold up to about 17.5 per cent. The rapidity of its absorption depends directly on the surface exposed by the gold, and account is taken of this fact in the usual statement of mill-men that more mercury is required to amalgamate fine gold than coarse. A large part of the mercury remaining in an amalgam after squeezing is held in solution in the gold, the remainder covering the particles in a film which assists in cementing the mass together.

1. *Absorption by the Plates.*—Important also is the absorption of the mercury by the plates. The investigation divides itself into two parts: the relative absorption of copper plates at different temperatures, and the relative absorption of different plates at ordinary temperature. For the former, sections of engravers' plate were used. They had each an area of 4.15 sq. in. and a thickness of 0.067 in., weighing approximately 40 grams. They were thoroughly amalgamated (using cyanide solution and sand) and kept submerged in mercury, the one at 100° C. (boiling water) and the other at 0° C. (melting ice) for three weeks, the rate of absorption being determined by removing the plates at intervals, freeing from the excess of mercury by rubbing and then weighing. It was not, of course, possible to free the plates entirely from adhering mercury; but by the use of care in the rubbing it was found possible to obtain fairly concordant results.

The curves A and B of Fig. 6 show that, as might have been expected, the rate of absorption is much faster at the higher temperatures, and the total amount absorbed is greater. Evidently from the direction of the curve the absorption is not complete, even at the end of three weeks. In the case of the plate held at 100° C. another factor enters. Little gritty crystals are formed on the plate which are strongly adherent,

though a few are removed by the rubbing. The plate, which was kept at 0° C., showed no indication of this, but in one instance a plate held at 20° C. did, to some extent. Apparently, then, the final limit of the absorption of mercury by copper plates is the formation of a solid solution or inter-metallic compound of a definite crystalline form.

The rate of absorption and total solubility depend on the temperature. One effect of the rise in temperature of plates that are in equilibrium can immediately be seen from an inspection of the diagram. On raising the temperature, more mercury will go into solution in the copper and also in the gold, and if more mercury is not fed to the battery, the plates will become dry and hard. The absorption is so slow that it will



A. Plain copper, absorption at 100° C. B. Plain copper, absorption at 0° C.

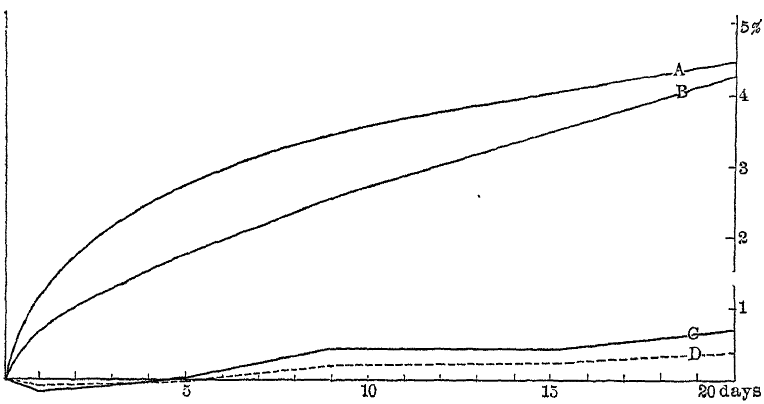
FIG. 6.—ABSORPTION OF MERCURY BY COPPER PLATE.

require several weeks to complete itself at the raised temperature; and, during that time, progressively varying conditions will have to be dealt with. Hence the importance of keeping the temperature constant, to avoid changes in the hardness of the amalgam.

To obtain the relative absorption-rates of plain and silvered copper plates, two similar plates were cut from electrolytic sheet-copper, and ground down to plane surfaces and rounded edges. Each had an area of 4.85 sq. in. and a thickness of 0.06 in. One was then silver-plated to a thickness of 0.005 in. Both plates were then amalgamated, kept submerged in mercury at 20° C., and the rate of absorption determined as before. The silver plate retains a little more mercury as an

adherent coat because of the greater specific gravity of the silver. The silver film somewhat restrains the diffusion of the mercury through it at first, but the ultimate effect differs little in the two cases.

The chief difficulty encountered in the use of plain copper plates is due to the fact that the small amount of copper which goes into solution in the mercury is very easily attacked by oxidizing agents and the salts which go into solution during the crushing of the ore, thus forming a tarnished film of copper salts on the surface of the mercury. Even the oxygen in solution in ordinary water produces this effect to an undesirable degree, but in the case of some ores containing certain



A. Plain copper. B. Silvered copper. C. Thin Muntz metal. D. Thick Muntz metal.

FIG. 7.—ABSORPTION OF MERCURY BY PLATES OF DIFFERENT METALS.

soluble salts the action frequently is so strong that no gold can be caught, as very little clean mercury surface is exposed. What these salts are is not known, except in a few cases. The data are not accurate enough to base generalizations upon. This difficulty is obviated by silver-plating the copper plates. The silver is soluble in the mercury to a very much less degree than is copper, and is also much less affected by the salts in question, probably not being at all acted on by the oxygen. The effect of the silver-plating on the absorption of the mercury by the copper is to restrain it at first, since the mercury has to diffuse through the silver. Eventually, however, the total amount absorbed is approximately the same, as is shown by the two curves, A and B, of Fig. 7.

The effect of the physical condition of the plates is also of

great importance. It will have been noticed by comparing Figs. 6 and 7 that the rolled engraver's plate at 100° C. absorbed more slowly than the electrolytic sheet-copper at 20° C. This must have been due to the difference in their structure, since the test was carefully repeated in each case.

2. *Use of Muntz Metal Plates.*—Muntz metal for battery-plates has been employed in Australia, and its use has been warmly commended,[34] but apparently has not spread much in this country or elsewhere. The advantages claimed for it are that it is cheaper, lasts longer, requires less attention, is easier to remove the amalgam from, and does not discolor. The solubility of mercury in Muntz metals was investigated, and plates for this purpose were kindly prepared by Mr. Frederick Maeulen (to whom my grateful acknowledgements are made). The metal had the composition: zinc 38.1 and copper 61.9 per cent. Plate C had an area of 5.11 sq. in. with a thickness of 0.0256 in.; while plate D had an area of 4.92 sq. in. and a thickness of 0.014 in. The plates were annealed and kept submerged in an excess of mercury. The negative reading at the end of the 24 h. is probably due to the mercury dissolving out zinc faster than it was itself absorbed. The solubility of mercury in Muntz metal is seen to be very low. The divergence of the curves emphasises the fact that the rate of absorption is proportional to the surface exposed, plate C having a much larger surface in proportion to its weight. The percentages are of weight, referred to the original weight of the plate. The curves also indicate that the absorption is very slow, and is not completed within the time during which observations were taken.

Hockin and Taylor have shown[35] that mercury dissolves the zinc out of brass. No copper could be detected in solution in the mercury at the end of the period. From these facts the reasons for the excellence of Muntz metal for plates become reasonably clear. Since no copper goes into solution in the mercury, no trouble is experienced from oxidizing agencies, and the necessity for silver-plating is removed in most cases. However, in working waste heaps[36] (which probably contained

[34] Rickard, *Stamp-Milling of Gold Ores*, pp. 179–182 (1901).

[35] *Journal of the Society of Telegraph Engineers*, vol. viii., p. 282 (1879).

[36] Rickard, *Stamp-Milling of Gold Ores*, p. 182 (1901).

copper sulphate) the Muntz metal plates coated badly; the small amount of zinc in the mercury producing even more undesirable results than would copper in such cases.

The Muntz metal consists almost entirely of an inter-metallic compound of copper and zinc. Mercury is very slightly soluble in this compound (perhaps not at all, for the slow gain in weight may be due to the slow replacement of one metal of the compound by the mercury). Gold is presumably also very slightly soluble in it; and this explains the slight adherence of the amalgam to these plates, for, beyond question, the strong adherence of amalgam to copper or silver plates is due to the slow diffusion of the gold into the metal. Roberts-Austen[37] has shown that gold diffuses into lead when brought into contact with it; and it is well known that alloys can be produced by pressing together finely divided metals that are capable of forming an alloy. This slight adherence is both an advantage and a defect; for it allows a closer cleaning up of the amalgam, and also makes it more frequently necessary. The thickness of the mercury film on the plates must also be important. The thickness of the film which a plate of a given inclination will hold depends on the specific gravity of the holding-surface. Hence, silvered plates will hold a thicker film, and Muntz metal a thinner. Plates well coated with gold amalgam hold the thickest film possible. Hence, excessive scraping of plates is to be avoided, since the gain in daily yield of amalgam is more than counterbalanced by the loss of "catching" power.

Since the Muntz metal is an inter-metallic compound of zinc and copper, there could be no possibility of any galvanic action between the zinc and copper to affect the process. Muntz metal plates present many desirable features, but apparently the demand has not been great enough to lead to their manufacture.

V. Investigation of Direct Changes Due to Temperature.

Whether variations of the temperature at which the process is conducted has any direct specific effect on amalgamation, has been the subject of much discussion. It will be impossible to refer here to all the statements made regarding this, but a few of the more significant must be mentioned. T. J. Grier[38] states

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[37] *Royal Society Proceedings*, vol. lxvii., p. 101 (1900).

[38] *Engineering and Mining Journal*, lxv., p. 126 (1888).

that when two batteries were run side by side on the same ore, the one at 50° F. and the other at from 60° to 70° F., considerably more gold was caught by the battery at the lower temperature. J. A. Church[39] cites a case where water from old mine-workings was used that had a temperature of 135° F. at the battery; the plates coated badly and the recovery was very poor. T. A. Rickard[40] says that at Black Hawk, Colo., the recovery is better in winter than in summer, and objects to the practice at the Britannia United, Ballarat, of warming the battery-water. Louis Janin[41] says that no advantage is gained by heating the battery-water. At the Golden Star mill, Supt. Read[42] changed a battery from 56° F. to 70° F. In 24 hr. the amalgam had nearly all scoured off. This was a silver plate; the copper was not affected. It is not stated whether all other conditions were kept constant, and it is hardly safe to comment on such an experiment unless all the conditions are known. J. A. Sanborn[43] says that the objection to an increase in temperature is that the plates were prepared for a lower temperature, but that the adjustment may be made for any reasonable temperature. He also says that the chief objection to hot battery-water is the resultant increased solubility of deleterious salts. J. H. Thierman[44] suggests that the heated water causes any grease which may be present to form more readily a film over the gold and mercury. A. von Dessauer[45] found that the plate-amalgam contained from 20 to 35 per cent. of gold in the summer time, but only from 7 to 10 per cent. in winter. Besides these factors, R. H. Richards[46] mentions two others, the attraction of mercury for gold and the cohesive power of the mercury. It should be noticed, however, that the only attraction mercury has for gold is the surface-tension pull, previously mentioned; and that the facilitation of the coalescence of globules of mercury by a rise of temperature is not due to an increase of cohesive power, but to a decrease of their surface-tension.

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[39] *Engineering and Mining Journal,* lxv., p. 158 (1888).
[40] *Stamp-Milling of Gold Ores,* pp. 125–6 (1901).
[41] *Mineral Industry,* iii., pp. 328 and 343 (1895).
[42] *Engineering and Mining Journal,* lxv., p. 126 (1888).
[43] *Engineering and Mining Journal,* lxv., p. 397 (1888).
[44] *Engineering and Mining Journal,* lxv., p. 247 (1888).
[45] *Engineering and Mining Journal,* lxv., p. 760 (1888).
[46] *Ore-Dressing,* vol. ii., p. 766 (1903).

The points in regard to the effect of temperature on the solubility of deleterious salts and resultant chemical reactions, their influence on "flouring," and the influence of rise of temperature on the surface-tension and viscosity of the mercury, have already been treated as fully as seems necessary.

To determine whether other physical constants are appreciably affected by temperature-changes, two lines of investigation were followed. The purpose of the first was to learn whether the plasticity of the mixture of liquid and solid, ordinarily known as "amalgam," varied to any sensible degree with ordinary ranges of temperature-change.

For this purpose an apparatus was designed and constructed, in which a pasty mass of amalgam was held in a cup that was rotated by a small electric motor. The cup was provided with sharp points on the inside, so that the amalgam was obliged to rotate with it. Submerged in the mass of amalgam was a stirrer, turning in a ball-bearing. The torsional pull on the stirrer, when the mass was rotated, was measured by a delicate helical spring. The whole apparatus was placed in a constant-temperature chamber, and numerous observations made over the range from 18° to 100° C. (60° to 212° F.). Both silver- and gold-amalgams of the consistency of thick cream were used; it was necessary to employ very finely divided metal in preparing the amalgam, in order to make it homogeneous. It was found that, as result of the increase of temperature, two conflicting changes tended to take place:—the amalgam tended to become harder from the increased absorption of the mercury by the gold, and softer from the decreased viscosity of the liquid mercury cementing the solid particles. The total change was very slight, and of the same order as the errors of observation due to the settling of the solid particles and the slight imperfections of the apparatus. A more detailed statement of the results, and a sketch of the apparatus, are therefore omitted. The investigation showed that the changes in plasticity induced by ordinary variations in temperature are negligible.

The purpose of the second investigation was to learn whether the adhesion of gold to mercury is sensibly affected by changes of temperature. To this end the inclination at which spheres of gold, silver and platinum, one-half an inch in diameter, will roll down an amalgamated copper-plate was observed. Two

sets of angles were measured: that at which the spheres would start to roll from a state of rest (column A, Table I.), and the angle at which they would continue to roll after being started by a touch from a camel's hair brush (column B, Table I.). The inclination was measured for the dry plate at ordinary temperature (20° C.), the amalgamated plate at the same temperature, and the amalgamated plate at 100° C. The gold and platinum spheres were perfect to within 0.001 in.; the silver sphere was not so perfect, and gave more irregular results. The angles recorded in Table I. are the average of several observations. The plate was somewhat roughened by the action of the mercury at the higher temperature; this is especially noticeable in the angles at which the balls would start themselves. The gold and silver balls were previously amalgamated for the determinations with the amalgamated plate, but the platinum ball, of course, was not. It is seen at once from an inspection of the results that the effect of variations of temperature on the adhesion of gold to mercury is also negligible.

TABLE I.—*Results of Adhesion-Tests of Gold, Silver and Platinum Balls.*

	Platinum.		Gold.		Silver.	
	A.	B.	A.	B.	A.	B.
Dry plates, 20°............	1° 28′	0° 28′	1° 09′	0° 32′	2° 17′	0° 48′
Wet plates, 20°............	1° 45′	0° 28′	1° 48′	1° 15′	3° 07′	1° 57′
Wet plates, 100°..........	1° 43′	0° 54′	2° 03′	1° 30′	....	1° 30′

These facts are so completely in accord with the data obtained by the metallurgical and chemical investigations, that there seems no escape from the conclusion that amalgamation of gold-ores is mainly, and above all, a simple mechanical process. Such chemical forces as enter into it are only slight, and are mainly detrimental, as previously outlined. The chief physical factor is the surface-tension of the mercury, which causes the mercury to cling in an adherent film to the plates employed, to wet the gold grains, and to draw them beneath its surface. It also acts detrimentally, tending to prevent the globules of mercury from coalescing again when they are very finely divided. The solution of the gold and silver by the mercury is

very slight indeed; that of the copper is more noteworthy, and gives opportunity for undesirable chemical actions. The absorption of the mercury by gold, silver and copper is large in amount. With gold, what is probably an inter-metallic compound is formed with excess of mercury at 100° (whether it can form at ordinary temperatures is not certainly known); but this is of no great importance in the ordinary application of the process. Some sort of molecular flow causes the grains of the amalgam to adhere strongly together on standing, and the slow diffusion of the gold into the plate causes the amalgam to adhere strongly to copper and silvered plates. According to Richards,[47] this diffusion does not extend to any great depth in the ordinary life of a plate.

The effect of changes in temperature on amalgamation is seen to be two-fold. The solubility of deleterious salts in the ore is, of course, increased by rise of temperature, their action is accelerated, and the solubility of copper in the mercury is increased, thus giving greater opportunity for these harmful reactions. But the most important effect is the disturbance of plate-equilibrium, resulting from changes of temperature. The thickness of the film of mercury which the plate will hold, and the absorption of mercury by the other metals concerned, are changed, and the mill-man then has to deal with varying conditions, a state of affairs to be avoided, if possible. From the physical standpoint, the temperature employed does not matter, so long as it is constant. From the chemical standpoint, a low temperature is better in most cases. But it should be mentioned, in passing, that both the viscosity and surface-tension of mercury are lessened by rise of temperature, the differences for ordinary ranges of temperature being slight. Under certain circumstances, where chemical action is negligible, it seems that it might be advisable to conduct the process at elevated temperatures, since the wetting of the grains by mercury is slightly facilitated by rise of temperature. The viscosity of mercury at 0° C. is 0.0169 dynes, and at 99°, 0.0123 dynes, per sq. centimeter.[48]

In the operation of the stamp-milling process the character and gold-content of the ore necessarily vary. If, in addition,

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[47] *Ore-Dressing*, vol. ii., p. 765 (1903).

[48] Koch, *Poggendorf's Annalen*, vol. xiv., p. 1 (1881).

the temperature, and with it all the foregoing factors, are allowed to vary, it becomes impossible to exert a proper control over the operation.

VI. Summary.

1. Gold absorbs mercury, forming a solid solution which may contain as much as 13 atomic per cent. of mercury. Beyond this, an inter-metallic compound containing gold or mercury in solution (or a second solid solution) is formed, which contains 17.5 atomic per cent. of mercury. Ordinary amalgam, which is not in a state of equilibrium, consists of one or both of the foregoing, usually the former, mixed with an excess of mercury which coats the particles and causes them to cohere.

2. Amalgamation is a physical process, the chemical actions involved being chiefly inimical (excepting those purposely induced). The gold grains are wetted by the mercury and sink beneath the surface of the mercury film on the plates; this is facilitated by feeding mercury to the stamp, so that the grains may be thoroughly wetted before coming in contact with the plates. The disadvantages of this procedure have already been discussed. The surface-tension of the mercury draws the gold beneath the surface, and holds it against the plate. By diffusion into the metal of the plates the amalgam often becomes strongly adherent. Silver-plating is useful, because it prevents the solution of the copper in the mercury, and, therefore, the harmful chemical reactions that result therefrom. Muntz metal plates exhibit the same effect, and, in addition, diffusion of amalgam into them is very slight, so that it is readily removed. Silvered plates will hold a thicker film of mercury than plain copper, and plates coated with gold amalgam a thicker film than either. This assists the "catching" of the gold.

3. Variations in temperature make themselves felt in slight changes of a number of factors rather than large changes in any one. According to the relative importance of these factors in each case the total effect may vary. The most important undesirable effects of raising the temperature are the increased solubility of harmful salts, and a corresponding increase of the precipitation of base metals into the mercury; this both hinders its proper action and leads to its loss. Rise of tempera-

ture also diminishes the surface-tension and viscosity of the mercury, which allows it to be more readily "floured." The force with which the gold is drawn beneath the mercury and held against the plate is also decreased. On the other hand, by an increase in temperature the wetting of the gold by the mercury and the "catching" of it by the plates is facilitated, as is the coalescing of the globules of mercury.

4. Increase of temperature causes increased absorption of mercury by the gold and by the plates. Changes in temperature cause changes in all the foregoing factors. The retaining of a constant temperature is therefore most favorable to successful working. A comparatively low temperature is better where the influence of soluble salts in the ore has to be considered (which is usually the case); but when this may be neglected, as high a temperature as can economically be maintained, without variation, is most favorable to successful amalgamation.

In conclusion, I desire to express my indebtedness to Prof. H. S. Munroe, who directed the investigation; and also to Dr. Wm. Campbell, who kindly took the micro-photographs, and aided in the metallurgical inquiry.

The Relative Merits of Large and Small Drilling-Machines in Development Work.

BY FREDERICK T. WILLIAMS, VICTOR, COLO.

(Bethlehem Meeting, February, 1906.)

THE purpose of this paper is to discuss the relative merits of the large $3\frac{1}{8}$-in. machine and the small $2\frac{1}{4}$-in. tappet machine in driving development-headings; and although the data here presented were obtained from cross-cut headings alone, experience has shown that the results are equally true in drifting, raising and winzing.

Recently we drove two parallel cross-cuts through the same formation, using a $3\frac{1}{8}$-in. machine at the breast of one cross-cut, and a $2\frac{1}{4}$-in. machine of the same make at the breast of the other. The results of this work afforded an ideal comparison, since in both cases the headings were advanced through rock practically of the same hardness and breaking-properties, the amount of sludging was equal, and there was no difference in the condition of the steel or the machines, the air-pressure or the experience of the operating crews.

Some operators in the Cripple Creek district contend that there is ground which cannot be handled with the small machine, the holes being too small to contain enough powder to "pull" the ground, etc. The results obtained in working the property of the Portland Gold Mining Co., however, show that the ground worked by them does not fall in this class. During a period of two years there have been driven, with the small machine, 4 miles and 308 ft. of development-headings, through a diversity of ground, including Pike's Peak granite (a coarsely porphyritic type of granite), highly indurated andesitic or phonolytic breccia, true massive andesite, trachytic phonolyte, tuffs, and along dikes of decomposed basalt and hard phonolyte. In every instance a satisfactory record was made.

The headings here described were driven through highly indurated andesitic breccia, having a hardness of from 5.2 to 7.2 and a sp. gr. of from 2.2 to 2.8. The action of the breccia under the drill was not materially different from that of ordi-

nary red granite. The breccia was not as free-drilling as granite, and sludge accumulated very rapidly after a shallow depth of hole had been gained, but it broke better than granite.

Aside from the usual work of setting-up, drilling, and loading, the machine-men or helpers mucked back, cleaning the floor 3 or 4 ft. back from the breast of muck in order to position the column properly. If the "lifters" acted properly at the previous firing, the muck was fairly-well thrown back from the breast; but if either missed fire or were exploded before the other holes, considerable muck was left at the breast which required much additional labor. The usual time needed to muck back was 1.25 hr., but this varied considerably. Flat steel 48 by 96 by $\frac{3}{8}$-in. sheets were used, from which to shovel the material. These sheets were placed in position 3 or 4 ft. back from the breast by the trammer, just before going off shift. The ground broke fine enough to require little or no sledging. A cubic foot of breccia in place will average 154 lb. in weight as compared with 90 lb. on the muck-pile, giving an average of 42 per cent. of void space. All the waste was trammed to the shaft, 800 ft. distant, and hoisted to the surface. No timber was used in either heading.

The following interesting summary of the results obtained by using both large and small machines has been prepared from the data given in Tables I. and II. Labor is the largest individual item. The wages of machine-men were $4 per shift, and the addition of the items given under the several heads of Table I. shows the total cost of labor performed in each heading. The cost of operating the machines per shift was $3.70 for the large machine and $1.85 for the small machine. These figures, which vary from month to month, include the cost of everything connected with the operation of the machines: engineer's wages, blacksmith-expense, new steel repairs to the machines, cost and repair of air-lines, etc. The cost of labor, per foot driven, by the large machine was $3.45, and by the small machine $2.56.

The cost of explosives, a detailed report of which is given in Table II., shows that the 40-per cent. dynamite costs $0.127 per lb.; the fuse, $0.0035 per ft.; and the caps, $0.007 each. These figures, which include freight, unloading, wages of the powder-man, and one-third of the wages of the store-keeper,

TABLE I.—*Development Report of the Portland Gold Mining Co. for 20 days Ending Oct. 16th, 1903.*

	Machine Men.	Machine Helpers.	Hand Miners.	Trammers.	Pipe and Trackmen.	Total Labor.	Cost of Labor per Foot.	Number of Machine Shifts Worked.	Cost of Operating Machines.	General Tramming Cost.	Explosives, Including Powder, Fuse, and Caps.	Cost of Explosives per Foot.	Cost of Pipe and Track.	Cost of Hoisting.	Cost of Supplies.	General Expense, Bosses, Assaying, Surveying, Etc.	Total Cost.	Cost per Foot.	Total Tons.	Cost per Ton.	Number of Feet Driven.	Feet per Shift.
Large machine ($3\frac{1}{8}''$). Cross-cut ($5.5' \times 7.5'$). 5 day run..............................	$40	$35	....	$22.13	$3.00	$100.13	$3 51	10	$37.00	$0.99	$66.22	$2.32	$11.69	$22.69	$0.79	$22.69	$262.20	$9 20	99.20	$2.64	28.5	2 85
8 day run............................	64	56	.. ..	31.13	3.75	154.88	3.33	16	59.20	1.42	90.20	1.93	19.07	32.57	1 14	32.57	391.05	8.41	142.40	2.75	46.5	2.91
7 day run............................	56	42	....	28.13	....	126.13	3.55	14	51.80	1.10	72.84	2.05	14.56	25.25	0.88	25.25	317.81	8.95	110.40	2.88	35.5	2.54
Averages and totals.,......	160	133	....	81.39	6.75	381.14	3.45	40	148.00	3.51	229.26	2.07	45.32	80.51	2.81	80.51	971.06	8.79	352.00	2.76	110.5	2.76
Small machine ($2\frac{1}{4}''$). Cross-cut ($4.5' \times 7.0'$). 5 day run..............................	40	......	$0.75	17.25	1.88	59.88	2.49	10	18.50	0.63	36.55	1.52	9.84	14.46	0.52	14.46	154.83	6.45	63.20	2.45	24.0	2.40
8 day run	64	......	....	31.88	3.75	99.63	2.49	16	29.60	1.09	53.01	1.32	16.40	24.89	0.87	24.89	250.38	6.26	108.80	2.30	40.0	2 50
7 day run........	56	......	....	30.38	3.75	90.13	2.69	14	25 90	1 10	36.71	1.09	13.74	25.08	0.88	25.08	218.62	6.53	109.60	1.99	33.5	2.39
Averages and totals.	160	......	0.75	79.51	9.38	249.64	2.56	40	74.00	2.82	126.27	1.29	39.98	64.43	2.27	64.43	623.83	6.40	281 60	2.22	97.5	2.44

represent the entire cost of the material, as laid down at the station for the machine-men. All fuse burned was in 7-ft. lengths. The number of feet of fuse burned and the number of caps used, per foot, are practically the same; but the cost of dynamite is $0.78 less per foot in the heading driven by the small machine than in that of the large machine.

TABLE II.—*Explosives—Detailed Report of the Portland Gold Mining Co. for 20 days Ending Oct. 16, 1903.*

	Lb. of Powder.	Lb. of Powder per Foot Driven.	Feet of Fuse.	Feet of Fuse per Foot Driven.	Number of Caps.	Number of Caps per Foot Driven.
Large machine ($3\frac{1}{8}''$). Cross-cut ($5.5' \times 7.5'$).						
5 day run	491	17.23	872	30.59	116	4.07
8 day run	669	14.39	1,179	25.35	158	3.39
7 day run	544	15.32	804	22.65	134	3.77
Averages and totals.	1,704	15.40	2,855	25.80	408	3.69
Small machine ($2\frac{1}{4}''$). Cross-cut ($3.5' \times 7'$).						
5 day run	264	11.00	672	28.00	96	4.00
8 day run	378	9.45	1,129	28.22	151	3.77
7 day run	262	7.82	742	22.15	120	3.58
Averages and totals.	904	9.27	2,543	26.08	367	3.76

The best record for a shift's run made by the large machine, was 4.08 ft., as compared with 2.96 ft. for the small machine. In drilling these rounds it was found that the large machine had made 3109.56 cu. in. of hole, and the small machine 971.10 cu. in. Comparing these figures with the cubic feet of ground pulled, 1 cu. in. of hole drilled by the large machine broke 0.053 cu. ft. of breast, while the small machine gave 0.097 cu. ft. This comparison shows that too much work was done by the big machine on the breast for the amount of ground broken.

Figs. 1 and 2 show the number of holes drilled, the degrees of pitch from the horizontal, the depth drilled by the starters, seconds and thirds, and the order of firing. The cost of coal before the boilers was $4.40 per ton. Ordinary cross or square bits were used, and all the steel was sharpened by machine. At each sharpening the steel lost from $\frac{1}{8}$ to $\frac{3}{8}$ in. in length. The general tramming-cost includes repairs to tram-cars, tram-tracks and the greasing of the cars.

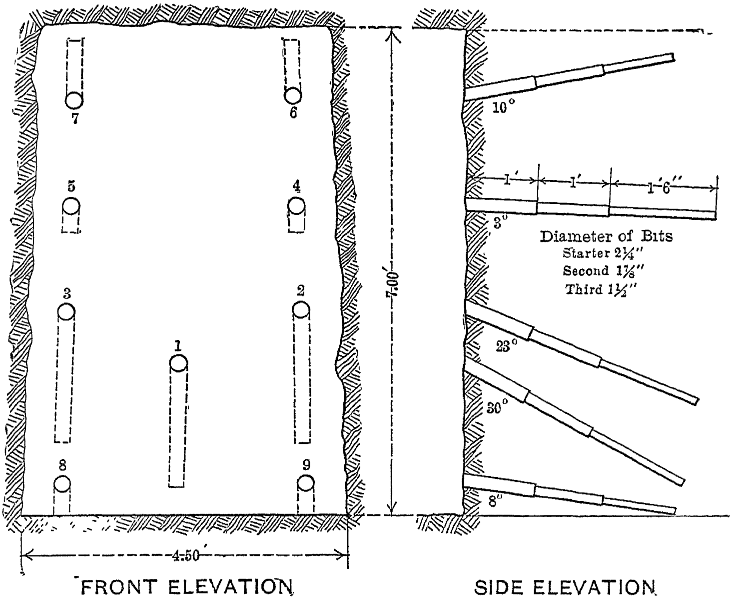

FIG. 1.—HOLES DRILLED IN BREAST BY SMALL ($2\frac{1}{4}$-IN.) MACHINE.

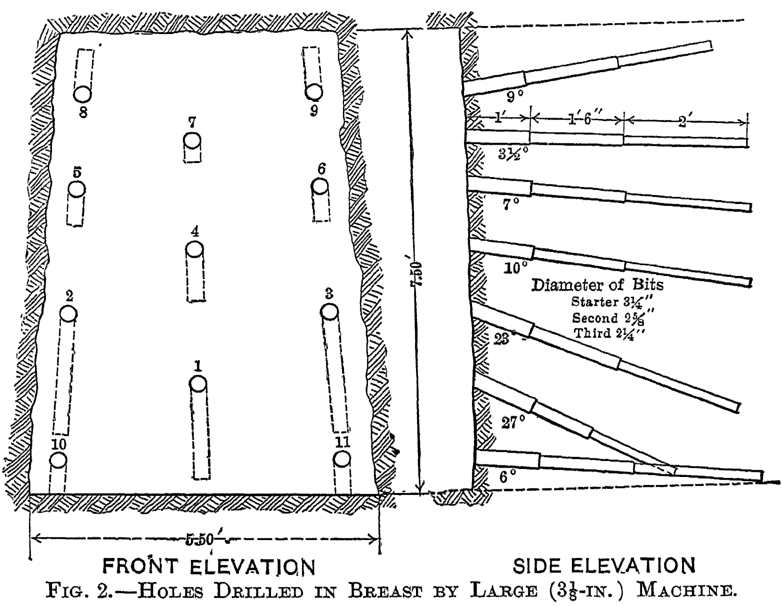

FIG. 2.—HOLES DRILLED IN BREAST BY LARGE ($3\frac{1}{8}$-IN.) MACHINE.

The cost of pipe and track is figured at $0.41 per ft., the 2-in. pipe costing, with connections, $0.10 per ft., the track,

together with the spikes, plates and ties, costing $0.31 per ft. Lumber cost $20 per thousand feet.

Hoisting cost $0.243 per ton, which includes all accounts that can be charged to the maintenance of the hoisting-engines;—such as wages of the engineers, wipers, top-men and cagers, repairs, cost of steam, cables and repairs to shaft. The hoist used is a 500-h.p. Webster, Camp & Lane, first-motion hoist, size 20 by 48 in., having a capacity for a maximum depth of 2,500 ft., using 5 by $\frac{3}{8}$ in. rope to hoist an unbalanced load of 8,000 lb. at an average speed of 1,500 ft. per min.

To supplies is charged the cost of picks and shovels. To general expense is charged the wages of foremen and shift bosses, assaying and surveying, pumping, lighting, including candles, office expense and general repairs on the surface.

The air was furnished by a 50-drill, cross-compound, Ingersoll-Sargent compressor, having steam-cylinders 24 by 44 by 48 in., and air-cylinders $22\frac{1}{4}$ by $38\frac{1}{4}$ by 48 in. The air-pressure at the receiver was 100 lb., and at the drills 85 lb. per sq. in.

The bore of the large machine cross-cut is 5.5 by 7.5 ft., that of the small machine is 4.5 by 7 ft. It is held that the increase of 1 ft. in width and 0.5 ft. in height of the large machine cross-cut over that of the small machine cross-cut does not facilitate mining-operations in any way.

The merits of the work done by the two machines may be briefly stated thus:—The use of the small machine saves 25 per cent. of the cost of labor necessary to operate a large machine foot for foot. The cost of operating a small machine is 50 per cent. less than that of operating a large machine, shift for shift. The general tramming-cost of the large machine cross-cut is lessened 20 per cent. by using a small machine. The cost of explosives per foot driven by the large machine can be reduced 37.7 per cent. by the use of the small machine. The cost of hoisting and general expense of the large machine cross-cut is lessened nearly 20 per cent. by using the small machine.

Greater speed, regardless of cost, can be obtained with the large machine, the small machine being from 10 to 20 per cent. slower. The cost of the large machine cross-cut was reduced 27 per cent. by using the small machine.

In conclusion, I wish to acknowledge my indebtedness to Mr. Frank L. Smale, Superintendent of the Portland property, for assistance received in gathering the data here presented.

Cost-Accounts of Gold-Mining Operations.

BY THOMAS H. SHELDON, VICTOR, COLO.

(Bethlehem Meeting, February, 1906.)

In the zeal for opening up new ore-bodies, or for extracting the ore from attractive bodies already opened up, we very often lose sight of the fact, that, after all, the operation of a mine is a business proposition, pure and simple, and, for the best working-results, should be treated upon a strict business basis. Of course, in every mine of consequence, a record is kept of expenditures and receipts, and such glittering generalities as "gross receipts," "net receipts," "mining expenses," and "per cent. profit," can be told to the cent; but does this record show economy of management, as compared either with the same record of other months, or with the record of other mines of the same class? Moreover, if such a record shows that the cost of mining is high, does it in any way enable the manager to put his finger on the leakage? Does it necessarily follow that a mine which makes a profit of, say, 40 per cent. on 25-dollar ore, is doing less economical mining than one which saves 60 per cent. on 80-dollar ore? Of course the figures in the latter case look the more attractive; yet when it comes to the point of saving everything which can be saved, and of cutting down expenses to the lowest possible cost of operation, the former mine is doubtless on the firmer and more economical financial basis. But as to the relative merits of the system of mining in the two cases, nothing could be decided without a basis of detailed comparison; if one system is more economical than the other, why is it so, and wherein does the advantage lie?

This can be shown only by keeping accurately the cost of each mining operation. And no matter how dissimilar two mines may be in character and operation, yet there are a few general heads common to all mining operations. In the first place, it is necessary to break the ore from the solid ground;

then it must be transported from the place where it is broken to the place of concentration (if any)—and this transportation generally includes two operations, underground tramming and hoisting; and, lastly, there is the cost of extracting the precious metal from the ore (and, generally, of transportation from the place of concentration to the place of extraction). In addition, there are accessory costs, such as timbering and pumping; and general costs, such as supervision, sampling and surveying; and such expenses as cannot be included in any one of the general heads above, yet are part of the cost of operation and should be apportioned among those heads. Then, too, there are the costs for exploration, and for equipment.

All expenses must be shown, not by themselves, but in regard to their direct effect upon the ore mined, and all can be shown in one way or another to be tributary to one of the general heads mentioned. For instance, it is a very interesting fact to know just how much it costs to run the boiler-room; yet the steam generated has no direct effect upon the ore, but only through the hoisting-, compressing- and pumping-engines; so that it is much more essential to know the value of the steam used, respectively, to break the ore, to hoist it, and to keep the mine dry enough to work in.

To show its relative efficiency, each department should have an account kept, wherein it is charged with all the debits which make up the cost of its operation, and credited with its proper contribution toward the whole general result. In other words, a double entry system of book-keeping is desired, where each operation, such as breaking ore, tramming, hoisting, etc., has its debit and credit account.

There are many methods of keeping these accounts. The ordinary book-keeping method is too clumsy, being too laborious in operation, and defeating the very object for which it was intended, by exacting too much time and trouble on the part of the management in inspecting and studying the various accounts in their relations to each other. The card-system has many adherents, and it is certainly an excellent plan for keeping the individual accounts which go to make up the cost-keeping system; but it is open to the same objection, in not showing at a glance the relations of the accounts to each other, to the whole result, and to the average cost, so that any leakage may

be readily discovered, located and properly remedied. And this is the need of the modern mine-manager.

This need has been met in the system now used by the Portland Gold Mining Co., Cripple Creek, Colo., which is applied in its simplest form—that is, the marketable product is only gold, which always has a market value of $20 per ounce. The mine is of large size, and there are numerous details to account for; therefore it is the purpose of this article to describe that system at some length, as being a typical and representative case. The figures shown herewith are those published in the company's annual report for the year 1902.

In brief, all the operating costs are represented upon one large sheet by co-ordinate methods; debits being figured along the line of abscissæ, and credits along the line of ordinates. Thus, every figure is viewed in two relations: as a credit, and as a debit. It is a credit to the account heading the vertical column under which it falls, and a debit to the account in the horizontal column opposite. The horizontal columns, then, show how the expenses accrue, and the vertical columns show how they are expended.

The accounts are divided into four main heads:—Milling, Plant and Development, Stoping, and Distributed Accounts. The last-named does not figure by itself in the operating costs, and its totals are not to be included with the other total debits; but these amounts are distributed or charged out among the other three heads: that is, the accounts which do not bear directly upon mining ore are charged out to those accounts which do. Thus, for example, the total cost of running the boiler-room is distributed or credited to the various purposes for which steam is used, such as hoisting and compressing. In the hoisting account, for example (the third line of the distributed accounts, Form 1): the amount which hoisting is indebted to the boiler-room is added to other debits chargeable to that process, showing the total cost of hoisting; this, then, is charged out or credited under the head of Stoping to the account of hoisting ore, and under the head of Development to the accounts of drifts, cross-cuts, etc., on the basis of the tonnage hoisted from stopes, drifts, cross-cuts, etc. Hence, all of the distributed accounts eventually find their way into the three main heads above,—accounts which bear directly upon mining the ore.

MILLING.—Since the Portland mill is not situated near the mine, its accounts are kept separate, and, under "MILLING," we have only two items—"total freight and treatment on ore sent to the Portland mill," and "total freight and treatment on ore shipped elsewhere." These two items are kept separate in order to compare the cost of treatment at the company's mill with that of custom-work.

PLANT AND DEVELOPMENT.—Under this head, the former includes only new buildings or the installation of new machinery—whatever may be considered a permanent improvement—and the latter includes underground operations which are of the nature of permanent equipment, such as a new shaft to facilitate hoisting, a new drift to open up a vein or system of veins, or a cross-cut to explore virgin territory. DEVELOPMENT is considered to embrace only those operations which could be cut off without materially affecting the production of ore. Under this head we find the sub-heads which embrace the total cost of running drifts and cross-cuts, of raising and sinking shafts and winzes. Thus it appears that everything under the head of PLANT AND DEVELOPMENT is an asset. All other costs are charged to *Stoping*, as this is considered to include all ordinary running expenses. These two headings need no further comment here.

STOPING.—This heading deserves further consideration to explain its sub-heads:—

Breaking Ore.—Under this term is included everything from drilling into the solid rock to delivering the ore into the chutes ready for the trammers. It embraces also the labor of the machine-drill men, hand-miners and muckers, the cost for running the machine-drills, the cost for explosives, etc.

Tramming.—The cost of getting the ore from the chutes to the stations of the shafts, including repairs to tracks and tram-cars.

Hoisting.—The cost of raising ore and waste to the surface, delivering the ore to the bins, and depositing the waste on the dump; includes the cost of running the hoisting-engines, labor of cagers and skip-tenders, repairs to the shaft and machinery, and surface-tramming and tram-tracks.

Timbering.—The cost of both labor and material used in keeping the mine timbered.

Sorting and Loading.—All expense of hand-concentration; the cost of tramming to the dump the waste picked from the ore, of loading the ore into the railroad-cars for shipment, and of keeping the ore-houses and bins in repair.

Pumping.—The cost of keeping the mine dry, including repairs to both pumps and lines.

Lighting.—Cost of electric lights and lines, and candles issued to the miners.

Assaying.—Including cost of mine-sampling.

Surveying.—Everything in the mine-engineering line.

Repairs.—Only those charges for general wear and tear which are not directly chargeable to any of the above heads—such, for instance, as painting the shaft-house.

General Expense.—Office accounts of both the Victor and the Colorado Springs offices; salaries of directors and officials, etc.

Insurance and Taxes.—Amounts paid for insurance and taxes on all mining property.

Litigation.—All legal expense connected with the mine.

Development Charged to Stoping.—Costs of those pieces of exploration-work which produce ore in their progress.

The other items need no comment here.

The general cost-sheet is supplemented by two auxiliary cost-sheets: one for Stoping and one for Development (Forms 3 and 4). On one or the other of these is found the cost of operation in detail, as well as the production and progress, of each working place in the mine. These sheets are not entirely separate and distinct from the cost-sheet, but are interdependent each upon the other, as will be more clearly shown later.

The scope of the cost-sheet can be most clearly and satisfactorily explained by referring to an actual example; so reference will be made to the one for December, 1902 (Form 1). This cost-sheet was published in the company's annual report for that year.

DISTRIBUTED ACCOUNTS.—Taking the first horizontal line,—the item of the boiler-room, we find this account indebted the sum of $99.56 to the machine-shop, $42.89 to the blacksmith-shop, $2.30 to the carpenter-shop, and also $464.92 to the supply-account, all of which represent labor and supplies used in repairing and maintaining the steam-lines, boilers and buildings, as well as the supplies, such as waste and oil, for opera-

tion; also to the pay-roll $1,073.25, the labor of the firemen; and to general office account $7.75, a voucher-account directly chargeable to making steam. Then we see the account of fuel debited with $5,063.10 by the general office, which sum represents vouchers covering the cost of the coal, delivered in the bins. This, then, appears as a lump sum charged to the boiler-room, showing as a debit to the boiler-room and a credit to the fuel account. Thus, summing up the debits of the boiler-room, the total cost of making steam appears to be $6,753.57.

The master mechanic's report shows that the steam was consumed in the proportions given in Table I.

TABLE I.—*Consumption of Steam.*

	Per Cent.
Pumps and dryers for washing waste rock in ore-house,	5.5
Pumps in mine,	18.4
Heating office-buildings and residences,	2.7
Compressors,	37.8
Hoisting-engines,	27.2
Dynamo-engine,	7.8
Steam-hammer (for sharpening drill steel),	0.6
	100.0

The cost of making steam, $6,753.57, is then charged to the above accounts in the above proportions, and the boiler-room credited to that extent, as shown in Table II.

TABLE II.—*Cost of Steam.*

Boiler-Room,	*Dr.*	*Boiler-Room,*	Per Cent.	*Cr.*
Machine-shop (repairs),	$99.36	Sorting and loading,	5.5	$360.64
Blacksmith-shop (repairs),	42.89	Pumping,	18.4	1,241.31
Carpenter-shop (repairs),	2.30	General expense,	2.7	180.32
Supplies,	464.92	Compressing	37.8	2,561.63
Fuel (delivered in bins),	5,063.10	Hoisting,	27.2	1,840.35
Pay-roll (firemen),	1,073.25	Electric-plant,	7.8	527.45
General office,	7.75	Machine-drills,	0.6	41.87
	$6,753.57			$6,753.57

In Table II., the first three credit items are ready to be added in as they stand, to form the total operating-costs, since they bear directly upon the ore account; they are now disposed of, as far as the distributed accounts go. The other items remain in the distributed accounts, in turn forming debits of other accounts.

The tabulation of other accounts is given in Table III.

TABLE III.—*Cost of Compressing, Hoisting, Electric Plant and Machine-Drilling.*

Compressing,	*Dr.*	*Compressing,*	*Cr.*
Boiler room, steam, ...	$2,561.63	Machine-drills, ...	$2,993.83
Machine-shop, repairs on compressors and air-lines, . .	22.54		
Supplies, oil, waste, packing,	130.66		
Pay-roll, engineers, ...	279.00		
	$2,993.83		$2,993.83

Hoisting,	*Dr.*	*Hoisting,*	Tons.	*Cr.*
Boiler-room, steam, ...	$1,840.35	Stopes (hoisting ore), ...	17,407	$4,118.60
Machine-shop, repairs on engines, ...	34.97	Drifts, ...	1,536	363.54
Blacksmith-shop, repairs on engines, ...	16.95	Cross-cuts, ...	2,439	577.20
Carpenter shop, repairs on engine-room, ...	19.75	Raises and winzes,	373	88.32
Supplies, lubricants, packing, ...	363.08	Shaft No. 1, . . .	66	3.31
Pay-roll, hoisting engineers,	2,886.44	Shaft No. 2, . . .	14	15.57
General office, ...	5.00			
Cost of hoisting 21,835 tons, ... or $0.2366 per ton.	$5,166 54		21,835	$5,166.54

Electric-Plant,	*Dr.*	*Electric-Plant,*	*Cr.*
Boiler-room, steam for generator, ...	$527.45	Lighting (mine and surface), .	$914.08
Machine-shop, electrician and repairs, ...	208.85		
Supplies, lamps, wire, etc., . .	177.78		
	$914.08		$914.08

Machine-Drills,	*Dr.*	*Machine-Drills,*	Shifts	*Cr.*
Boiler-room, steam-hammer,	$41.87	Breaking ore, . . .	1,575	$3,648.71
Compressing, air for machines, ...	2,993.83	Drifts, ...	$267\frac{3}{4}$	620.42
Machine-shop, repairs on machines, ...	203.58	Cross-cuts, ...	365	845.37
Blacksmith-shop, sharpening steel, ...	1,728.13	Raises and winzes, .	66	151.70
Supplies, repair parts and new steel, ...	599.88	Shaft No. 2, ...	10	23.16
		Shaft No. 3, ...	120	277.93
Cost of operating, ... $2,403\frac{3}{4}$ shifts, or $2.3166 per shift.	$5,567.29		$2,403\frac{3}{4}$	$5,567.29

An examination of the credit account of the boiler-room, together with the other debit accounts, will show that all the boiler-room expense has now been distributed among the various operating costs. In like manner all of the distributed ac-

counts eventually find their way into the operating accounts, and are there absorbed by the various items under that head.

The final result of the cost-sheet is now merely a question of cross-footing the various operating accounts, and balancing with the line called *Total Cost of Mining*. This gives, of course, the amounts in the total debits column, and summing these up gives the total cost of operating the mine.

The last three vertical columns of the cost-sheet speak for themselves; the figures are obtained merely by dividing the total debits by the tonnage gross, tonnage shipped, and ounces of fine gold produced. These figures, of course, are invaluable for showing the efficiency of the operations of mining, and may be said to be the pulse by which the condition of the mine may be felt at once. Of these three columns, the last should be the criterion, although the cost per ton shipped is more often referred to. But as the same gross result may be obtained by shipping a large tonnage of low-grade ore or a small tonnage of high-grade ore, yet the costs and profits will be vastly different in the two cases; so the cost of producing an ounce of fine gold, regardless of tonnage, is the real criterion. For, after all, the object of running a gold-mine is not to produce gold-ore so much as fine gold.

It will be noticed that on the cost-sheet shown herewith, there are really two cost-sheets in one, side by side; one for the current month, and one for the current year to date—in this case a period of six months, of which the current month is the last. This latter cost-sheet is kept side by side with the other, in order that the current month may be compared at a glance with the rest of that year preceding, and also that the results of a year's run may be readily obtained. These figures are not worked out separately, but are carried forward from month to month.

The basis of the distribution into the heads of *Stopes*, *Drifts*, *Cross-cuts*, etc., is found in the auxiliary-sheets mentioned before. Thus the number of shifts worked with machine-drills to break ore is found from the stope-sheet to be 1,575$\frac{3}{8}$, and the number required to run drifts shows on the development-sheet to be 260$\frac{7}{8}$, for cross-cuts, 311$\frac{1}{2}$, for raises and winzes, 56, and for sinking, 70. This gives a total of 2,273$\frac{3}{4}$ shifts. Then on the development-sheet it will be seen there were 130 shifts

worked with large machines; and as these require just twice as much air as the small ones, 130 should be added to 2,273¾, giving an equivalent of 2,403¾ shifts. Now from the cost-sheet the total cost for running machine-drills appears to be $5,567.29, or $2.3166 per shift. This constant, multiplied by the number of shifts for each heading, gives the cost of machines for that place. And thus also for the other costs on the auxiliary sheets—the explosives and timber being charged out as used, and the other costs being apportioned pro rata according to production.

The results of the cost-sheet, such as the total cost of mining, cost of milling, per ounce and per ton, together with other items of general interest, are plotted into graphic curves. These form very interesting and instructive diagrams, one of which is shown in Fig. 1. The upper line represents the average value of the ore in dollars per ton shipped; the next line below shows the total cost of producing and marketing a ton of ore; the difference between it and the line above representing of course the net profits per ton. The second line is the sum of the two smooth lines below—the cost of mining, and the cost of treatment. In the same way the next broken line below, the total cost of producing an ounce of fine gold, is composed of the two costs represented in the lower broken lines—the cost of mining an ounce of gold, and of extracting that amount of the precious metal from the rock. The dotted line at the bottom represents the cost of mining a ton of crude rock. The figures forming the basis of these curves are taken from the cost-sheet.

It will be well to note that the general upward tendency of the costs during October, November and December is due to the installation of new and expensive machinery; while the downward tendency of the mining-costs in December is the beginning of the effect of this machinery. The upward movement of the treatment costs is on account of the higher grade of ore being treated.

At the mine the different sets of curves are shown in different colors of ink. Other diagrams also are kept, such as the cost and progress of development work, tonnage handled and shifts worked.

Comparison with the new cost-sheet form published herewith (form No. 2), which has recently supplanted the old form

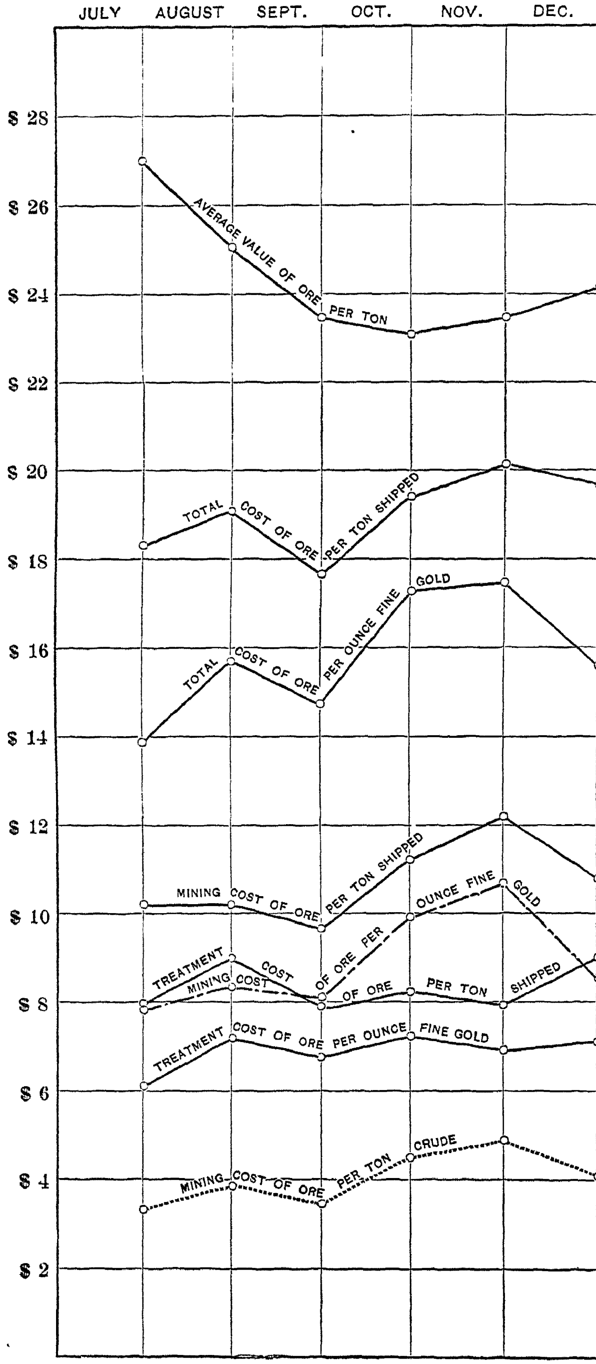

FIG. 1.—DIAGRAM OF OPERATING-COSTS OF THE PORTLAND GOLD MINING CO.

just explained, shows the new system differing from the old only in the greater amount of detail shown, especially in the distributed accounts; the principle of the system remains unchanged. By the addition of a number of lines to the distributed accounts, and corresponding vertical columns, detail can be shown to any extent desired. Greater detail is also obtained by the addition of the four vertical columns headed Shafts Nos. 1, 2, 3, and Lowell. These four columns balance by themselves with the total debits column, but are separate from the rest of the debits. The total debits column, then, is balanced in two different ways: with the four working-shafts, and with the rest of the cost-sheet. Take, for example, the hoisting account. The cost of the hoisting operation for each shaft can be ascertained with great accuracy, also the amount of repairs put upon each engine, thus showing at a glance the relative efficiency of each hoisting-plant. But in the total debits column only the grand total of all the hoisting expense is entered; this is distributed and charged out as before shown in the old cost-sheet. As the shafts, except the Lowell, are all connected underground, their mining-expenses cannot always be kept separate, but merge one into the other, and such items as breaking ore and tramming, which do not fall naturally into divisions by shafts, are charged out to the various shafts on the basis of the tonnage hoisted from each.

The addition of the account called "*Invoice*" will also be noticed. As such accounts as supplies, timber, explosives, and fuel are charged out as used and debited as bought, the debits and credits do not balance on the old cost-sheet; by charging this difference to the account of *Invoice* on the new form, the accounts balance, and the invoice account, carried forward from month to month, shows the amount of these articles in stock.

The two cost-sheets are both shown in this article, to illustrate how the same system of the co-ordinate method of cost-keeping may be made either general or particular. In the old form the system is a very general one, and could be condensed and generalized even further; while the second form goes into considerable detail, without departing in any essential respect from the main principle.

Such a system of cost-keeping may be objected to on the ground of involving too much "red tape," and therefore being

too expensive in its operation. In reply it is maintained that any system which accomplishes its purpose in keeping accurate record of the cost of operating is cheap at any cost. But the present system is not nearly so cumbersome and expensive as it doubtless looks at first glance. It has now been kept up at the Portland office for nearly three years, with the employment of the same office-force as was employed for the same length of time previous to its installation.

To explain just how each one of the basic figures is obtained would be going into a great deal of unnecessary detail, a large part of which would be too local to be of general benefit, as the operation of this system is made to fit the material at hand, and is largely a product of growth. However, a sketch of the every-day labor, as practised at the Portland, may be instructive.

There are three general heads under which the entire cost of production may be grouped:—first, *pay-roll;* second, *voucher-account*, for supplies and all other expenses which appear on the cost-sheet under the head of "General Office;" and third, *freight and treatment charges.* Theoretically, the sum of these three should be the total cost of operation, and on the new cost-sheet it actually is, when the invoice account is considered. The freight and treatment charge is taken in a lump sum directly from the ore-book, and requires no further comment here; thus the whole of the cost-sheet has its derivation in the distribution of the other two accounts, the pay-roll and the general office.

The distribution of the pay-roll is accomplished by each shift-boss or foreman making out a report of each day's work upon a regular form (forms Nos. 5, 6, 7, 8, 9, 10 and 11) which shows the number of shifts or hours worked upon any particular job under his supervision. These reports are all handed in to the office at the close of the shift, and the next day are compiled. This then shows the daily cost of each piece of work—the cost of operating the machine-drills, the cost of tramming to the shaft, the labor-cost of timbering, or of putting up a new building. Then at the end of the month it is simply a matter of addition to get the labor-cost of each piece of work done, the total of all, of course, balancing with the total pay-roll. But the cost thus found is not yet ready to go

into the cost-sheet as it stands—there are certain additions to be made to it. For instance, repairs to the machine-shop might necessitate the assistance of a carpenter; the cost of his services is then taken from the carpenter-shop account and charged to the machine-shop; but as it benefits the whole mechanical department, each charge in that department has to stand its share of it. And so, too, with the wages of supervision—this cannot be charged to any particular job, but must be apportioned among all the pieces of work under that supervision. The way these charges are apportioned appears in Table IV.

In Table IV. the first column represents the actual labor-costs as taken directly from the daily reports. Then, grouped under the sub-heading "*Distributed*" are the items, general in nature, which must be distributed over these amounts to make the true costs. The second column shows the distribution, pro rata, of this amount. The figures in the third column are the sum of those in the first two, and are the amounts which are now to be entered directly upon the cost-sheet.

For example, the last line of the first division is the item \$7.88 charged against the carpenter-shop. This charge, then, is placed directly against the carpenter-shop, and appears under the sub-heading "*Distributed.*" Then this \$7.88, with other similar items, make the \$199.08 which is to be distributed over the items forming the total of \$1,502.39, or at the rate of about 13.25 per cent. Each amount in the second column is about that percentage of the corresponding figure in the first, and their sum forms the amounts in the third column, which then appear on the cost-sheet.

The distribution of the general office column of the cost-sheet comes from the voucher-record, upon which is kept a distribution of each account as it is vouchered. This form is not shown here, being merely a book similar to that generally used in connection with the voucher-system. These accounts have to be treated the same as those of the pay-roll, as just explained, before entering upon the cost-sheet. However, the supplies-, explosives-, and lumber- and timber-accounts are further distributed. No supplies are issued except upon written order to the store-keeper (form 12), signed by the foreman of the department, and stating the specific use of the article; this forms the basis of the storekeeper's distribution of supplies issued.

TABLE IV.—*Distributed Costs of Machine-Shop and Carpenter-Shop.*

Machine-Shop

	Pay-roll.	Distributed.	Total.
Machine-drills	$188.50	$15.08	$203.58
Machinery (new)	1,881.88	152.52	2,034.40
Hoisting	32.38	2.59	34.97
Compressing	20.87	1.67	22.54
Sorting and loading	22.50	1.80	24.30
Electric plant	193.38	15.47	208.85
Blacksmith-shop	65.88	....	65.88
Pumping	61.38	4.91	66.29
Tramming	116.25	9.30	125.55
Supplies	233.12	18.65	251.77
Boiler-room	92.00	7.36	99.36
No. 2 shaft	212.75	17.02	229·77
Buildings	14.87	1.19	16.06
General expense	25.00	2.00	27.00
Carpenter-shop	7.88	....	7.88
	3,168.64	249.56	3,418.20
Distributed:			
Lumber and timber	....	1.20	....
Blacksmith-shop	....	1.94	....
Supplies	....	92.92	....
General office	....	153.50	....
Total	....	$249.56	....

Carpenter-Shop.

	Pay-roll.	Distributed.	Total.
Lumber and timber	$485.43	$62.00	$547.43
Cribbing	101.88	13.00	114.88
Machinery	51.00	6.90	57.90
Buildings	633.70	85.00	718.70
General expense	5.25	0.70	5.95
Sorting and loading	85.38	11.53	96.91
Assaying and sampling	4.00	0.55	4.55
Boiler room	2.00	0.30	2.30
Hoisting	17.25	2.50	19.75
Repairs	6.00	0.80	6.80
Surveying	9.50	1.40	10.90
No. 2 shaft	15.00	2.00	17.00
No. 3 shaft	86.00	12.40	98.40
Total	$1,502.39	$199.08	$1,701.47
Foreman	155.00	....	....
	$1,657.39	....	....
Distributed:			
Foreman	....	$155.00	....
Machine-shop	....	7.88	....
Supplies	....	13.30	....
General office	....	22.90	....
	....	$199.08	....

Explosives are issued by the powder-man only upon an ammunition-order from the machine-drill man, stating the heading where it is to be used (form 13). From these data the powder-man makes up his monthly report (form 14). The lumber and timber used are distributed by the carpenter-foreman, who keeps account of all the lumber used in repairs and construction about the surface, as well as the underground timbers. The latter account is still further subdivided by the head timber-man, to whom is reported daily the amount of lumber and timber used in each heading by each timber-man (form 15). This gives him data for his monthly timber-record (form 16). At the end of the month the columns are added and figured at a price sufficient to cover cost of framing and handling, as well as of purchase. The amount of fuel used is reported by the master mechanic, who weighs it as it is used under the boilers. The master mechanic also reports the distribution of the steam generated, based upon the horse-power of the engines consuming it.

The basis for the auxiliary sheets is the shift-bosses' distribution-sheet (form 17), one of which is used for each stope, drift, cross-cut, etc., in the mine, and upon which the shift-boss enters the distribution of his shift. The tonnage from each place is obtained by counting the number of trammers' tags (form 18) from that place, and multiplying by the car-constant (0.7 ton); these tags the trammers place upon the cars as they leave them at the stations, and the top-men remove them and turn them into the office. Of the other costs upon the auxiliary sheets, the explosives and lumber and timber are charged directly from the powder-man's and head timber-man's reports; the cost for machines is the debit against machine-drills on the cost-sheet, distributed on the basis of the number of drill-shifts worked; the other costs are taken from the cost-sheet and distributed on the basis of tonnage. The column for feet of progress on the development-sheet is filled in by the surveyor.

Upon the above, it will doubtless be commented that this throws a good deal of clerical work upon the shift-bosses and foremen—which is very true; but there is nothing complicated in what these men are asked to do—nothing that the man with average intelligence enough to be a good foreman cannot do in

20 or 30 minutes at the close of the shift. And this expenditure of time more than pays for itself in giving the foreman a line on his work—a review of the work of the day and a basis for the plans of the next. It is merely a question of devoting about that much time to it every day. By obtaining the co-operation of the foremen, the system is not nearly so expensive as may be imagined.

No progressive and business-like mine-operator will question the efficacy of keeping close watch upon his cost of production—of having some system which will show at a glance at regular intervals where a saving might be effected, and which will compare the efficiency of present working with past. The only question is as to how this can best be accomplished.

While it is not claimed that the system above described is the best that could be devised for all cases, yet it is claimed that it is the best for the conditions here at hand; and it is also claimed that the excellent showing made by the Portland mine in recent times is entirely due to close supervision guided by the cost-sheet.

The forms employed by the Portland Mining Company are given in full on the pages which follow. For the purpose of this publication, they have been much reduced in size, and provided with explanatory notes which do not appear upon the sheets used in the office. I trust that, thus presented, together with the preceding comments, they will enable mine-managers to estimate intelligently the value of the system, and the extent to which, with suitable modification to adapt it to local conditions, it may be employed with advantage.

In conclusion, I wish to disclaim credit for any originality in the system described, except in a few minor details, such as the graphical diagrams of the results. The credit for installing the cost-sheet at the Portland mine (in 1902), and for working out its general ideas, belongs to Mr. J. R. Finlay, then manager; in this labor he was ably assisted by Mr. L. F. Curtis, the purchasing agent, who put the system upon a practical working-basis, and who has since modified it to suit the growing needs of the mine. To these gentlemen, together with Mr. F. M. Kurie, the present manager, I wish gratefully to acknowledge my indebtedness.

List of Forms.

1. *General Cost-Sheet.*
2. *General Cost-Sheet, with Distributed Accounts.*
3. *Stope-Report.*
4. *Development-Report.*
5. *Shift-Report.*
6. *Ore-House Report.*
7. *Surface-Report.*
8. *Carpenter-Shop Report.*
9. *Distribution of Labor in Machinery Department.*
10. *Blacksmith-Shop Report.*
11. *Mechanical Department.*
12. *Order on Storekeeper.*
13. *Ammunition-Order.*
14. *Powder-Man's Report.*
15. *Timberman's Report.*
16. *Timber-Report.*
17. *Shift-Bosses' Distribution-Sheet.*
18. *Trammer's Tag.*

20,425 Tons of Ore Stoped for Month of December, 1902. 121,677.755 Tons Stoped for 6
7,837.19 Tons of Ore Shipped for Month of December, 1902. 47,649.26 Tons Shipped

		Boiler-Room.		Compressing.		Hoisting.		Electric Plant.		Tramming.	
		1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos
OPERATING ACCOUNTS. Stoping.	Breaking ore										
	Tramming									518.92	2,852.93
	Timbering										
	Hoisting					4,118.60	25,956 40				
	Sorting [illegible]	[illegible]	[illegible]								
	Pumping	1,241 31	19,517.78								
	Lighting							914.08	4,526.02		
	Assaying										
	Surveying										
	Repairs										
	Gen. expense, Victor	180.32	712.20								
	Gen. expense, Colo. Sp.										
	Insurance and taxes										
	Litigation										
	Total	1,782 27	22,089.84			4,118.60	25,956.40	914.08	4,526.02	518.92	2,852.93
Development.	Drifts					363.54	2,787.97			45 21	495 61
	Cross-cuts					577.20	1,710 96			72.27	228.41
	Raises					88.32	390.72			10.85	58.59
	Shaft No. 1					3.31	302 87			0.41	50 90
	Shaft No. 2					15.57	15.57			1.91	1.91
	Shaft No. 3		323.84								
	Cribbing										
	Buildings										
	Machinery										
Plant and											
	Total		323 84			1,047.94	5,208.09			130.65	835.42
	Total cost mining	1,782.27	22,413.68			5,166,54	31,164.49	914.08	4,526.02	649.57	3,688.35
Milling.	Portland mill										
	Other freight—treat										

Distributed

DISTRIBUTED ACCOUNTS.	Boiler-room										
	Compressing	2,561.63									
	Hoisting	1,840.35									
	Electric plant	527.45									
	Tramming										
	Timbering										
	Machine-shop										
	B. S. shop										
	Carpenter-shop										
	Machine-drills	41.87		2,993.83							
	Explosives										
	Lumber and timber										
	Supplies										
	Surface expense										
	Fuel										
	General office	4,971.30		2,993.83							
		6,753.57		2,993.83		5,166.54		914.08		649.57	

EXPLANATION.—All accounts are debited in the horizontal columns opposite
The "Distributed Accounts" do not figure by themselves in
out among the various operating accounts under the heads

Company.—General Cost-Sheet.

NOTE.—Form 1, the original size of which is 21 by 43 5 in , covers 4 pages of this paper. The left half of the sheet is given on this and opposite page, and the right half on the two following pages.—R. W. R.

Months. 9,888 Ounces Fine Gold for Month of December, 1902.
for 6 Months. 56,996.137 Ounces Fine Gold for 6 Months.

Timbering.		Machine-Shop.		B. S. Shop.		Carpenter-Shop.		Machine-Drills.		Explosives.	
1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos	1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos.
					1,214.68			3,648.71	20,728.88	3,135.00	18,307.30
149.22	9,427.37										
		24.30	280.03	16.91	106 66	96.91	584.25				1.37
		66.29	505.63	7.26	89.64						
					2.28	4.55	4.55				
						10.90	16.40				
						6.80	86.80				
		27.00	27.00			5.95	87.70				
											1.13
149.22	9,427.37	117.59	812.66	24.17	1,413.26	125.11	779.70	3,648.71	20,728.88	3,135.00	18,309.80
	1,156.52				541.42			620.42	5,385.18	846.48	7,950 04
								845.37	3,613.45	984.58	4,804.73
	617.89							151 70	1,745.34	142.18	1,338.61
					101.98		55.50		962.40		457.23
	495.04	229.77	229.77	21.33	21.33	17.00	21.66	23.16	23.16	16.31	48.58
			55.13	4.53	196.39	98.40	350.66	277.93	1,853.83	248.15	1,564.30
						114.88	1,615.72				
		16.06	410.18		7.53	718.70	2,960 82				
		2,034.40	6,010.71	166.08	340.83	57.90	741.17				
	2,269.45	2,280.23	6,705.79	191.94	1,209.48	1,006.88	5,745.53	1,918.58	13,027.86	2,237.70	16,163.49
149.22	11,696.82	2,397.82	7,518.45	216.11	2,622.74	1,131.99	6,525.23	5,567.29	33,756.34	5,372.70	34,473.29

Accounts.

		99.36		42.89		2.30					
		22.54									
		34.97		16.95		19.75					
		208.85									
		125.55		40.19							
				12.08							
				1.94							
		65.88									
		7.88									
		203 58		1,728.13							
						547.43					
		251.77									
				2.94							
		1,020 38		1,845.12		569.48					
149.22		3,418 20		2,061.23		1,701.47		5,567.29		5,372.70	

their names and credited in the vertical columns below their names.
the operating costs, but their amounts are distributed or charged
of "Stoping," "Plant and Development" and "Milling."

Form 1.—*Portland Gold Mining Company.*—

Lumber and Timber.		Supplies.		Surface Expense.		Fuel.		C. S. Office		Pay-Roll.	
1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos
....	....	43.11	570 19	....	....	....	....	....	....	10,023.74	60,459.22
....	....	....	80.00	....	....	....	....	....	....	4,723.75	30,377.82
4,690.45	11,097.18	....	39.36	....	....	....	....	....	....	4,270 50	21,391.26
....	....	....	....	....	....	....	....	....	....	....	....
5.36	30.76	129 47	676.01	145.88	313.11	....	....	....	....	5,893 23	35,469.89
....	21.02	35.13	1,864.06	....	....	....	....	....	....	291 00	5,220.64
0.80	0.80	370 95	1,706.10	....	....	....	....	....	....	....	....
....	2 20	4 06	16.63	....	....	....	....	....	....	....	373.88
....	3.12	59.52	100 54	....	2 33	....	....	....	....	96.38	547.51
18.80	117.60	6.55	43 37	155.87	1,287.52	....	....	....	....	....	....
....	....	8.38	27.53	29 12	187.71	....	....	....	1,214.14	186.00	1,091.00
....	....	....	....	....	....	....	....	3,028.51	22,099.98	....	....
....	....	....	....	....	....	....	....	....	15,261.49	....	....
....	....	....	....	....	....	....	....	3,290.00	6,974.89	....?..	....
....	....	....	....	....	....	....	....	....	....	....	....
4,715.41	11,272.68	657.17	5,123 79	430 87	1,790.67	....	....	6,318.51	45,550.50	25,484 90	154,931.22
35.85	35.85	....	94.97	....	....	....	....	....	....	2,147.18	18,556.99
....	....	....	....	....	....	....	....	....	....	3,089.03	13,230.85
99.82	249.10	....	....	....	....	....	....	....	....	808 13	7,747 56
10.28	563 41	....	184.87	....	....	....	....	....	....	48.00	3,235 49
5.16	41.88	18 32	53.72	....	....	....	....	....	....	131.81	223 81
407.30	2,025.79	32 06	590 51	....	....	....	162 00	....	....	1,987 10	10,431 78
185.60	2,264.54	....	72.50	....	130.64	....	....	....	....	....	....
881.82	2,918.27	1,545.33	2,122 99		270.85	....	....	....	....	....	96.00
66.60	843 62	2,278.20	6 201.65	14 50	330 49	....	....	....	8,038.97	....	51.25
....	....	....	....	....	....	....	....	....	....	....	....
....	....	....	....	....	....	....	....	....	....	....	....
1,692.43	8,942.46	3,873.91	9,324.21	14.50	731 98	..	162.00		8,038 97	8,211.25	53.573 73
6,407 84	20,215.14	4,531.08	14,448.00	445.37	[illegible]		[illegible]	[illegible]	33,589 47	33,696.15	208,504.95
....	....	....	....	....	....	....	....	....	....	....	....
....	....	....	....	....	....	....	....	....	....	....	....
....	....	....	....	....	....	....	....	....	....	....	....
....	....	....	....	....	....	....	....	....	....	....	....

Distributed

....	....	464 92	....	....	....	5,063.10	....	....	....	1,073.25	....
....	....	130.66	....	....	....	....	....	....	....	279.00	....
....	....	363.08	....	....	....	....	....	....	....	2,886.44	....
....	....	177.78	....	....	....	....	....	....	....	....	....
8.16	....	182 17	..	212 50	..	..	..	..		81 00	.
....	....	137.11	.	..	....	..	....		..	..	
1.20	....	92.92	....	....	....	....	....	....	....	3,168 64	....
....	....	92.92	....	....	....	....	....	....	....	1,660.69	....
....	....	13.30	....	....	....	....	....	....	....	1,657.39	....
....	....	599.88	....	....	....	....	....	....	....	....	....
....	....	....	....	....	....	....	....	....	....	192.75	....
....	....	....	....	320.89	....	....	....	....	....	....	....
....	....	....	....	75.12	....	....	....	....	....	70 00	....
....	....	11.18	....	....	....	....	....	....	....	990.76	..
....	....	....	....	....	....	....	....	....	....	....	....
9 36	....	2,265.95	....	608.51	....	....	....	....	....	12,059 92	....
6,417.20	....	6,797.03	....	1,053.88	....	5,063.10	....	....	....	45,756.07	....

General Cost-Sheet.—Continued.

Gen Office.		Total Debits		Cost per Ton Crude		Cost per Ton Shipped.		Cost per Ounce Gold.		
1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos.	1 Mo.	6 Mos.	
285.00	1,887.96	17,135.56	103,168 23	0 8390	0.84.79	2.1864	2.1651	1.7328	1.8101	Breaking ore.
140.00	636.67	5,382.67	33,947 42	0 2635	0.27 90	0.6868	0 7124	0 5443	0 5956	Tramming
……	399.96	9,110.17	42,355.13	0.4460	0.34.81	1.1624	0 8889	0.9213	0.7431	Timbering
……	100.00	4,118.60	26,056 40	0.2016	0.21.41	0.5255	0 5470	0.4165	0.4572	Hoisting.
191.50	1,276.50	6,864.50	40,598.44	0.3361	0.33.37	0.8759	0 8520	0.6942	0.7123	Sorting and loading.
……	100.00	1,640.99	27,318.77	0.0803	0.22.45	0.2094	0 5733	0.1659	0.4793	Pumping.
……	859.23	1,285.83	7,092.15	0 0630	0 05 83	0.1641	0.1488	0.1300	0.1244	Lighting.
615.77	4,390.75	624.38	4,790.29	[illegible]	[illegible]	0.0797	0.1005	0.0631	0.0840	Assaying.
211.39	1,226.04	378.19	1,895 94	0 0185	0 01.56	0 0483	0.0398	0.0383	0.0333	Surveying.
……	108.80	218.02	1,644 09	0 0107	0 01.35	0 0278	0 0345	0.0220	0 0288	Repairs.
3,948.64	9,033.80	4,455.41	12,381 08	0.2181	0 10.18	0 5686	0 2598	0 4506	0.2172	General expense, Victor.
0.35	84.94	3,028.86	22,186.05	0.1483	0.18.23	0 3865	0 4656	0.3063	0 3892	General expense, Colo. Sp
……	……	……	15,261.49	……	0.12.54	……	0 3203	……	0.2678	Insurance and taxes.
100.00	300.00	3,390 00	7,274.89	[illegible]	[illegible]	0.4326	0 1527	0.3428	0.1277	Litigation.
……	……	6,114.05	24,812.50	[illegible]	[illegible]	0.7801	0 5207	0.6183	0.4353	Development charged to Stoping—2,571 tons.
5,492.65	20,404.65	63,747 23	370,782 87	3.1210	3.04.73	8.13.41	7 78 14	6 4464	6.50 53	Total.
……	283.59	4,058 68	37,298.14	……	……	0.5179	0.7828	0.4104	0.6544	Drifts.
……	121.07	5,568.45	23,709.47	……	……	0.7105	0.4976	0.5631	0.4158	Cross-cuts.
……	110.31	1,301.00	12,258.12	……	……	0.1660	0 2572	0 1315	0 2150	Raises
……	45.50	62.00	5,904.65	……	……	0.0079	0 1239	0 0063	0.1035	Shaft No. 1.
……	……	480.34	1,176 43			0.0618	0.0247	0 0486	0.0205	Shaft No 2.
……	50 25	3,055.47	[illegible]	……	……	0 3900	0 3589	0 3089	0 3008	Shaft No. 3.
……	……	300.48	4,083.40	……	……	0 0382	0 0857	0.0304	0 0716	Cribbing.
962.54	4,386.96	4,124.45	13,173.60	……	……	0.5263	0.2765	0.4171	0.2310	Buildings.
2,686.73	33,652.88	7,304.41	56,204.57	……	……	0.9320	1.1795	0.7386	0.9861	Machinery
……	……	26,255.28	170,912.86	……	……	3.3501	3.5868	2.6549	2 9987	Total.
……	……	6,114.05	24,812.50	……	……	0.7801	0.5207	0.6183	0 4353	Less Dev. charged to Stoping.
3,649.27	38,650.56	20,141.23	146,100.36	……	……	2.5700	3.0661	2 0366	2 5634	Total.
9,141.92	59,055.21	83,888.46	516,883.23	……	……	10.7041	10.8475	8.4830	9.0687	Total cost mining.
……	……	[illegible]	[illegible]	……	……	[illegible]	7.4695	……	……	Portland mill (oper'n).
……	……	[illegible]	[illegible]	……	……	[illegible]	11.7405	……	……	Other freight—treat.
……	……	70,363.42	396,539.03	……	……	8.9781	8.3219	7.1153	6.9573	
……	……	154,251.88	913,922.26	……	……	19.6822	19.1694	15.5983	16.0260	

Accounts.

7.75	……	6,753 57	……	……	……	……	……	……	……	Boiler-room
……	……	2,993.83	……	……	……	……	……	……	……	Compressing.
5.00	……	5,166 54	……	……	……	……	……	……	……	Hoisting, 21,836 tons.
……	……	914.08	……	……	……	……	……	……	……	Electric plant.
……	……	649.57	……	……	……	……	……	……	……	Tramming, 22,349 tons.
……	……	149.22	……	……	……	……	……	……	……	Timbering.
153.50	……	3,418.20	……	……	……	……	……	……	……	Machine-shop
241 74	……	2,061.23	……	……	……	……	……	……	……	Blacksmith-shop.
22.90	……	1,701 47	……	……	……	……	……	……	……	Carpenter-shop.
……	……	5,567 29	……	……	……	……	……	……	……	Mach.-drills, 2,403¾ shifts.
5,252.63	……	5,445.38	……	……	……	……	……	……	……	Explosives.
3,542.96	……	4,411.28	……	……	……	……	……	……	……	Lumber and timber
3,735.81	……	4,132 70	……	……	……	……	……	……	……	Supplies.
49.00	……	1,053 88	……	……	……	……	……	……	……	Surface expense.
5,063.10	……	5,063.10	……	……	……	……	……	……	……	Fuel.
18,074 39	……	49,481 34	……	……	……	……	……	……	……	General office.
27,216.31	……	……	……	……	……	……	……	……	……	

FORM 2.—*The Portland Gold Mining Co.—General Cost-Sheet.*

Ore Stoped, Month of ....190 .., ...Tons. Ore Shipped, Month of.........190..., ..Tons. Fine Gold, Month of. ... ..190. ., . Ounces. Development, Month of.........190 . . .. Feet
Ore Stoped, ...Months, ...Tons. Ore Shipped, ...Month, ...Tons. Fine Gold, ...Months, ... Ounces. Development, ...Months, . . .Feet.

		Boiler-Room.		Com-press'g.		Mach'e-Drills.		Hoist-ing.		Auxil-iary Hoists.		Electric Plant.		Tram-ming.		Timber ing.		Ore-Houses.		Waste Haul'ge System		Surface Ex-pense.		Carpen-ter-Shop.		Black-smith-Shop.		Mach'e-Shop.		Fuel.	
		1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	.. Mos	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	.. Mos.	1 Mo.	.. Mos.	1 Mo.	.. Mos.	1 Mo.	... Mos.	1 Mo.	. Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	.. Mos	1 Mo.	.. Mos	1 Mo.	.. Mos.
OPERATING ACCOUNTS. [illegible]	Breaking ore																														
	Trammming																														
	Timbering																														
	Hoisting																														
	Sorting and loading																														
	Pumping																														
	Lighting																														
	Assaying and sampling																														
	Surveying																														
	Repairs																														
	General expense—Victor																														
	General expense—Colo Sp.																														
	Insurance and Taxes																														
	Litigation																														
	Total stoping																														
Development.	Drifts																														
	Cross cuts																														
	Raises and winzes																														
	Shaft No. 1																														
	No. 2.																														
	No. 3.																														
	Lowell																														
	Total development																														
	Total cost mining																														
Milling	Portland mill.																														
	Other freight and treatment																														
	Total freight and treatment																														
	Total mining and milling																														
Perm. Equip.	Buildings.																														
	Machinery																														
	Cribbing.																														
	Head works																														
	Rolling stock and rail																														
	Pipe lines																														
	Tools and appliances																														
	Total plant account																														
	Total operating cost																														

NOTE.—Form 2, the original size of which is 21 x 43.5 in., covers 4 pages of this paper The left half of the sheet is given on this and the opposite page, and the right half on the two following pages.—R. W. R

Boiler-room; Operation
Repairs and maintenance
Water-supply
Compressing: Operation
Repairs and maintenance
Air-lines
Machine-drills:
Repairs
Sharpening steel
New steel
Drill sharpener
Hoisting;
Repairs and maintenance
Top trammers
Shaft repairs and maintenance
Auxiliary hoists: Operation
Repairs and maintenance
Electric Plant: Dyn. & Engine
Repairs and maintenance
Lamps
Lines
Tramming: Apportioned
Timbering; Apportioned
Ore-houses: Rep. and main'ce
Sorting
Loading
Tramming waste
Washers—Operation
Washers—Rep. and maint'ce
Waste haulage system:
Rope haulage—Operation
Rep. & main.
L. B. Elevator—Operation
Rep. & main.
Belt conveyors—Operation
Rep. & main.
Surface expense
Carpenter-shop
Blacksmith-shop
Machine-shop
Fuel
Lumber and timber
Explosives
Supplies
Total distributed accounts
Totals
Invoice
Balance

EXPLANATION.—All accounts are debited in the horizontal columns opposite their names and credited in the vertical columns under their names. The "Distributed Accounts" do not figure by themselves in the operating costs, but their amounts are distributed or charged out among the various Operating Accounts under the heads of "Stoping," "Development," "Milling" and "Permanent Equipment." The columns headed "Shaft 1, 2, 3, and Lowell" show subdivision of total costs by shafts

DISTRIBUTED ACCOUNTS.

FORM 2.—*The Portland Gold Mining Co.—General Cost-Sheet.—Continued.*

Lumber, Timber.		Explo-sives.		Sup-plies.		Pay-Roll.		General Office.		Shaft No. 1.		Shaft No. 2.		Shaft No. 3.		Lowell Shaft.		Total Debits.		Cost per Ton Crude.		Cost per Ton Shipp'd.		Cost per Ounce Gold.			Tons.	
1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.	1 Mo.	... Mos.		1 Mo.	... Mos.
																										Breaking ore		
																										Tramming		
																										Timbering		
																										Hoisting		
																										Sorting and loading		
																										Pumping		
																										Lighting		
																										Assaying and sampling		
																										Surveying		
																										Repairs		
																										General expense—Victor		
																										General expense—C. S.		
																										Insurance and taxes		
																										Litigation		
																										Total stoping		
																										Feet. 1 Mo. / ... Mos.		
																										Drifts		
																										Cross-cuts		
																										Raises and winzes		
																										Shaft No. 1		
																										No. 2		
																										No. 3		
																										Lowell		
																										Total developm't.		
																										Total cost mining		
																										Portland mill		
																										Other freight and treatment		
																										Total freight and treatment		
																										Total mining and milling		
																										Buildings		
																										Machinery		
																										Cribbing		
																										Head works		
																										Rolling stock and rail		
																										Pipe lines		
																										Tools and appliances		
																										Total plant account		
																										Total operating cost		
																										Net		
																										Gross		

DISTRIBUTED ACCOUNTS.

Boiler-room: Operation	
Repairs and maintenance	
Water-supply	
Compressing: Operation	
Repairs and maintenance	
Air-lines	
Machine-drills:	
Repairs	
Sharpening steel	
New steel	
Drill sharpener	
Hoisting:	
Repairs and maintenance	
Top trammers	
Shaft repairs and maintenance	
Auxiliary hoists: Operation	
Repairs and maintenance	
Electric plant: Dyn'o and engine	
Repairs and maintenance	
Lamps	
Lines	
Tramming: Apportioned	
Timbering: Apportioned	
Ore-houses: Repairs and main'nce	
Sorting	
Loading	
Tramming waste	
Washers—Operation	
Washers—Repairs and main'nce	
Waste haulage system	
Rope haulage—Operation	
Repairs and maintenance	
L. B elevator—Operation	
Repairs and maintenance	
Belt conveyors—Operation	
Repairs and maintenance	
Surface expense	
Carpenter-shop	
Blacksmith-shop	
Machine-shop	
Fuel	
Lumber and timber	
Explosives	
Supplies	
Total distributed accounts	
Totals	
Invoice	
Balance	

FORM 3.—*The Portland Gold Mining Co.*—

For month ending

		Machine-Men.	Hand-Miners.	Shovelers.	Trammers.	Pipe-and Track-men	Timbermen.	Timber Helpers.	Total Labor.	Machines (Large or Small).	Number Machine Shifts Worked.
200	Portland	40.00	...	...	27.00	3.00	14.00	12.00	96.00	S.	10
400	Bobtail	142.00	...	3.00	81.00	4.13	42.00	36.00	308.13	.	$35\frac{1}{2}$
...	No. 4 Lee	...	...	66.00	54.00	2.25	14.00	12.00	148.25	...	...
...	No. 5 Captain	453.50	154.50	739.13	547.88	28.13	361.82	420.00	2,704.96	...	$113\frac{3}{8}$
...	No. 7 Captain	588.00	6.00	12.00	145.50	10.13	128.62	111.76	1,002 01	...	147
...	No. 9 Captain	648 00	4.13	30.58	66.50	16.50	106.32	91.13	963 16	..	162
500	No. 3 Captain	164.00	1.13	165.38	134.25	24.38	357.00	325.88	1,172 02	.	41
...	No. 4 Captain	320 00	.	12.38	104.35	1.50	47.25	49.50	534.98	...	80
...	No. 5 Captain	252 00	3 00	60.00	111.38	3.75	25.38	23.25	478.76	...	63
...	No. 7 Captain	64.00	...	84.00	90.75	3.38	10.50	9.00	261.63	...	16
...	No. 9 Captain	392.00	...	70.13	186.75	2. 25	40.25	27.00	718 38	...	98
600	Burns	40.00	...	15.75	55.50	...	14.00	12.00	137.25	...	10
...	Old No. 2	...	69.00	...	69.00	0.75	3.50	3.00	145.25	...	...
...	No. 2 Stope	236.00	...	109.50	132.00	4.50	59.50	51.00	592.50	...	59
...	North Diamond	...	...	...	...	0.38	3.50	3.00	6.88	...	...
...	Bobtail	316.00	...	4 50	113.25	9.38	56.00	54.00	553.13	...	79
...	No. 3 Captain	212.00	1.13	18.38	135.00	8.25	17.50	15.00	407.26	...	53
...	No. 4 Captain	32.00	...	...	74.25	...	...	...	106 25	...	8
...	No. 7 Captain	420.00	...	18.00	124.50	...	63.00	51.00	676.50	...	105
...	No. 8 Captain	12.00	...	...	25.00	2.25	...	...	39.25	...	3
...	No. 2 Hidden Treas.	...	...	15.00	28.50	...	3.50	3.00	50.00	...	...
700	No. 2 Stope	384.00	118.50	178.50	198.00	4.13	87.50	51.00	1,021.63	...	96
...	Rose	...	...	...	...	...	45.50	57.00	102.50	...	...
...	No. 1 Hidden Treas.	...	105.00	84.00	139.50	1.88	154.00	21.00	505.38	...	...
...	No. 2 Hidden Treas.	...	...	3.00	3.00	...	17.50	21.00	44.50	...	...
...	No. 3 Lee	...	...	...	6.00	1.13	...	...	7.13	...	...
...	No. 4 Lee	4.00	...	39.00	48.00	0.75	...	...	91.75	...	1
800	Bobtail	...	...	...	48.38	0.38	...	...	48.76	...	...
...	No. 1 Hidden Treas.	...	...	39.00	64.75	3.75	...	...	107.50	...	...
...	No. 3 Hidden Treas	...	...	3.75	10.13	3.38	...	...	17.26	...	...
900	No. 3 Hidden Treas.	624.00	82.50	109.50	345.00	5.63	136.50	138.00	1,441.13	...	156
...	No. 4 Lee	202.00	3.00	30.00	81.00	6.38	35.00	30.00	387.38	...	$50\frac{1}{2}$
...	East Fork	396.00	...	...	84.00	1.50	...	...	481.50	...	99
1,000	No. 3 Hidden Treas.	160.00	174.38	270.00	552.75	4.13	192.50	181.50	1,535.26	...	40
...	No. 4 Lee	200.00	...	31.50	69.75	4.88	77.00	66.00	449.13	...	50
1,100	East Fork	...	...	...	9.00	...	7.00	6.00	22.00	...	...
		6,301.50	722.27	2,211.98	3,965.62	162.83	2,120.14	1,881.02	17,365.36		1,575$\frac{3}{8}$

Stope Report (Size of Original Sheet, 18 by 19.25 in.).

December 31, 1902.

Cost for Machines.	General Tramming Cost.	Explosives.	Lumber and Timber.	Hoisting.	Supplies.	General Expense: Bosses, Assaying, Surveying, etc.	Total Cost.	Tons Ore.	Tons Waste Hoisted.	Tons Waste Dumped in Old Stopes.	Total Tons.	Cost per Ton.
23.16	2 44	32.04	....	19.87	3.09	37.90	214.50	56	28	... .	84	2.55
82.22	7.55	82.83	37.26	61 28	9 58	117.18	706.03	225	34	. ...	259	2.72
....	8 42	4.45	....	68.62	10 68	130.76	371.18	266	24	....	290	1.29
262.58	60.37	3.06	1,475.90	491.20	76.53	936.77	6,110.24	1,918	158	... .	2,076	2.94
		98.87										
340.47	31.72	193.61	38.16	258 13	40.19	492.08	2,396.37	1,080	11	....	1,091	2.20
375.21	30.78	345.20	36.30	250.35	39.01	477 55	2 517.56	1,055	3	.....	1,058	2.38
94.96	21.59	57.19	1,003.96	117.40	27.33	334 79	2 829 24	389	107	246	742	3.81
185.29	11.05	246.12	19.56	82.57	13.99	171.50	1,265.06	343	6	31	380	3.33
145.92	17.65	104.01	16.10	133.92	22.37	273.83	1,192.56	500	66	41	607	1.96
37.06	13.55	27 61	13.60	103.63	17.18	210.35	684.61	323	115	28	466	1.47
226.98	20 39	216.02	16.40	159.24	25 83	316.15	1,699.39	639	34	28	701	2.42
23.16	5.23	15.23	7.64	42.58	6 62	81.17	318 88	170	10	....	180	1.77
... .	6.04	19 86	....	49.21	7.46	93 80	321 62	176	32	....	208	1.54
136.65	19.14	154.87	68.59	155.69	24.25	296.88	1,448.57	613	45	....	658	2.20
....	....	1.45	....	....	....	.. ..	8.33	....	... .	. . ....	....	....
182.97	11.43	132.97	39.15	92.99	14.50	177.50	1,204.64	386	7	. .....	393	3.06
122.75	12.57	119.97	23.00	102 45	15 93	195 19	999.12	389	44	....	433	2.31
18.53	7.10	17.08	3.25	57.73	8 99	110.23	329.16	181	63	....	244	1.35
243.18	13.65	216.87	26.58	113.35	17.30	211.83	1,519.26	412	67	....	479	3.17
6.95	1.81	39.71	....	14.67	2.29	28.11	132.79	49	13	....	62	2.14
....	2.44	....	....	19.87	3.09	37.90	113.30	66	18	.. ..	84	1.35
222.35	31.20	179.39	411.90	241.10	39 51	484.18	2,631.26	970	49	54	1,073	2.45
....	....	....	177.33	....	....	....	279.83	....	....	....	....	.. ..
....	18.03	12.55	107.20	146 67	22.83	279.51	1,092.17	494	126	....	620	1.76
....	1.08	9.43	39.70	8.75	1.35	16.74	121.55	15	22	....	37	3.29
....	2.14	2.12	....	17.51	2 70	33.16	64.76	67	7	....	74	0.88
2.32	4.68	14.06	....	38.09	5.92	72.64	229.46	148	13	....	161	1.42
....	4.92	0.88	....	40.22	6.24	76.43	177.45	143	27	....	170	1.04
... .	5.03	. .. .	....	40 93	6 37	78.01	237.84	138	35	....	173	1.37
. . ..	3.32	0.61	....	26 97	4.17	51.48	103 81	101	13	....	114	0.91
361.30	41.27	325.90	240.10	335.74	52 25	640.20	3,437.89	1,195	224	. ...	1,419	2.42
116.96	15.77	89.80	63.42	128.23	19.96	244 46	1,065.98	513	29	....	542	1.97
229.29	12.86	171.92	....	100.08	16 29	199.61	1,211.55	422	1	19	442	2.74
92.64	58.79	105.22	760.50	478.41	74.53	912.14	4,017.49	1,640	382	....	2,022	1.98
115.81	14.36	94.10	63.55	116.65	18 16	222.66	1,094.42	461	32	....	493	2.22
....	0.55	....	1.30	4.50	0.68	8.53	37 56	19	....	....	19	1.97
3,648.71	518.92	3,135.00	4,690.45	4,118.60	657.17	8,051.22	42,185.43	15,562	1,845	447	17,854	2.36

FORM 4.—*The Portland Gold Mining Company*

For Month Ending

	Drifts.	Machine-Men.	Machine Helpers.	Hand-Miners.	Shovelers.	Trammers.	Pipe- and Track-men.	Timbermen.	Timber Helpers.	Total Labor.
200	Burns.........S.	56.00		3.00		54.00	22.50	24 50	21.00	181.00
400	Tidal Wave.........S.	80.00				42.00	10.94	0.44		133.38
500	Bobtail.........N.	196 00				50.00	12 38			358.38
	No. 5 Captain.........S.	36 00				121.75	2.63			60.38
600	Burns.........SW.	40.00				22 50	4.13			66.63
	No 8 Captain.........S.	8.00				3.75	0.75			12.50
800	No. 2 Captain.........S.	112.00				83.25	10.88			206.13
	No. 3 Captain.........N.					1.50	0.75			2.25
	No. [illegible] S.	[illegible]				[illegible]	1.88			97.13
	No. [illegible] N.						0.38			0.38
900	No. [illegible] N.	[illegible]				[illegible]	2.25			126.13
	No. [illegible] S.	[illegible]				[illegible]	5.63			126.88
1,000	No. 4 Captain.........N.	40.00				15.75	25.50			81.25
	No. 3 Hidden Treasure.. N.	108 50	24 50	27.00		342.50	14.25			516.75
1,100	No. 3 Hidden Treasure...N.	52 00				27.38	4.25			83.63
	No. 3 Hidden Treasure....S.	[illegible]				39.00	3.38			94.38
		991.50	24.50	30.00		932.76	122.48	24.94	21.00	2,147.18
	Cross-cuts.									
Adit	No. 4 Captain.........W.	44.00				32.62	2.63			79.25
	No. 4 Captain.........E.	56.00				37.87	4.50			98.37
500	No. 7 Captain.........NE.	212.00				181.88	16.13			410.01
600	Burns.........E.	[illegible]				[illegible]	2.25			73.25
	No. [illegible] Captain.........E.	[illegible]			[illegible]	[illegible]	20.25			1,160.13
700	No. 2 Hidden Treasure.........E.	[illegible]				[illegible]	12.00			321.50
1,100	From No. 1 Shaft.........S.			[illegible]		[illegible]	3.00			30.00
	No. 1 Shaft to No. 2.........N.	[illegible]	[illegible]			63.00	6.38	Contract. 267 50		432.38
	No. 2 Shaft to No. 1.........S.	64.00	49.00			42.00	1.13	245.00		401.13
	No. 1 Cross-cut from No. 2 D.........W.	56.00				23 63	3.38			83.01
		1,138.00	94.50	18.00	365.25	889.13	71.65	512.50		3,089.03
	Raises.									
400	No. 1 Tidal Wave.........	96.00	33.25	168.00		45.38	4.50	122.50	51.00	520.63
	No. 2 Tidal Wave.........	128.00				70.50	4.50	45.50	39 00	287.50
		224.00	33 25	168.00		115.88	9.00	168.00	90.00	808.13
	Shaft.									
600	No. 2 Station.........	40.00				3.00	21.00	1.31	Contract. 39.50	131.81
1,100	Burns' Station.........					6.00			42.00	48.00
	No. 3 Shaft.........							42.50	1,725.60	1,768.10
		40 00				9.00	21.00	43.81	1,807.10	1,947.91

Development-Report (Size of Original Sheet, 18 by 19.25 in.).

December 31, 1902.

Machines (Large or Small).	Number Machine Shifts Worked.	Cost for Machines.	General Tramming Cost.	Explosives.	Lumber and Timber.	Hoisting.	Supplies.	General Expense. Bosses, Assaying, Surveying, etc.	Total Cost.	Tons Ore.	Tons Waste Hoisted.	Tons Waste Dumped in Old Stopes.	Total Tons.	No. Feet.	Cost per Foot.
S.	14	32.43	3.76	42.30	35.85	30.53	......	58.43	384.30	3	126	......	129	26½	14.50
......	20	46.32	3.01	74.26	......	24.62	......	46.74	328.33	......	104	......	104	33	9 95
......	49	113.49	10.70	189.67	......	87.10	......	166.12	925.46	174	194	......	368	92	10.06
......	9	20.84	1.12	32.50	......	4.73	......	17.37	136.94	16	4	19	39	15	9.13
......	10	23.16	0.98	21.15	......	8.05	......	15.16	135 13	5	29	......	34	18	7.56
......	2	4.63	0.55	5.99	......	4.50	......	8 53	36.70	11	8	......	19	......	......
......	28	64.85	4.82	70.30	......	39.29	......	74.85	460.24	116	50	......	166	29½	15.60
......	......	......	0.20	......	......	1.66	......	3.16	7.27	6	1	......	7	......	......
......	12¾	29.53	2.42	48.25	......	19.65	......	37.59	234.57	48	35	......	83	34	6.89
......	......	......	......	......	......	......	......	......	0.38	......	......	......	......	......	......
......	20	46.32	2.30	38.85	......	18.70	......	35.69	267.99	15	64	......	79	31	8.64
......	20	46.32	3.68	72.76	......	30.06	......	57.17	336.87	106	21	......	127	37	9.10
......	10	23.16	1.20	33.98	......	9.70	......	18.64	167.93	......	41	......	41	16	10.49
L.	7	32 43	5.61	110.93	......	45.67	......	87.17	875.28	3	190	......	193	61½	14.23
S.	20⅛	76.72													
	13														
......	13	30.11	2.16	52.77	......	17.51	......	33.48	219.66	......	74	......	74	25	8.78
......	13	30.11	2.69	52.77	......	21.77	......	41.69	243.41	41	51	......	92	29	8.39
......	260⅞	620.42	45.20	846.48	35.85	363 54	......	701 79	4,760.46	544	992	[illegible]9	1 555	[illegible]	[illegible]
......	11	25.48	1.91	61.51	......	15.62	......	29.68	213.45	......	66	......	66	29½	7.23
......	14	32.42	2.79	57.06	......	22.72	......	43.27	256.63	......	96	......	96	31½	8.14
......	53	122.74	11.34	225.14	......	81.17	......	175.92	1,026.32	......	343	47	390	107½	9.55
......	11	25.48	1.44	50.38	......	11.83	......	22.42	184.80	......	50	......	50	27	6.85
......	109	252.44	40.32	177.10	......	328.34	......	625.99	2,584.32	1,074	313	......	1,387	14	184.59
......	44	101.92	7.69	165.83	......	62.39	......	119.39	778.72	5	259	......	264	94	8.28
......	......	......	1.04	2.47	......	8.52	......	16.11	58 14	......	36	......	36	......	......
L.	13	60.22	1.77	82.48	......	14.43	......	27.48	676.66	......	61	......	61	53½	12.65
L.	12½	57.90													
S.	2	4.63	2.22	108.06	......	17.98	......	34.43	698.15	......	76	......	76	49	14.24
L.	14	64.85													
L.	14	64.85													
S.	14	32.43	1.75	54.55	......	14.20	......	27.16	213.10	......	60	......	60	21½	9.91
......	311½	845.36	72.27	984.58	......	577.20	......	1,121.85	6,690.29	1,079	1,360	47	2,486	427½	15.65
S.	14½	33.58													
L.	9½	44.00	5.93	77.00	67.69	48.32	......	92.01	889.16	49	155	......	204	72	12.35
......	32	74.12	4.92	65.14	32.13	40.00	......	76.44	580.25	1	168	......	169	48	12.09
......	56	151.70	10.85	142.14	99.82	88.32	......	168.45	1,469.41	50	323	......	373	120	12.24
S.	10	23.16	1.91	16.31	......	3.31	18.32	29.69	224.51	......	66	......	66	......	......
......	......	......	0.41	......	......	15.57	......	6.32	70.30	......	14	......	14	......	......
L.	60	277.93	......	248.15	......	......	32.06	......	2,326.24	......	944	......	944	94½	24.72
......	70	301.09	2.32	264.46	......	18.88	50.38	36.01	2,621.05	......	1,024	......	1,024	94½	2 24.7

FORM 5.—*The Portland Gold Mining Co.—Shift Report.**

Victor, Colo.,............................190...

Shaft No........ ....M. Shift.

Shifts.	@		
		Shift-boss.	In stopes.
		Machine-men.	
		Miners.	
		Muckers.	
		Stope sweepers.	
		Trammers.	
		Timbermen.	
		Timber helpers.	
		Nippers.	General.
		Track- and pipe-men.	
		Cagers.	
		Topmen.	
		Car greasers.	
		Powdermen.	
		Machine-men.	Development.
		Miners.	
		Trammers.	
		Timbermen.	
		Timber helpers.	
		Total Shifts.	

....

Shift-Boss.

* Size of original sheet, 8.5 by 5.5 in.

FORM 6.—*The Portland Gold Mining Co.—Ore-House Report.**

Portland, Colo.,..........................190...

No. 1.		Occupation.	No. 2	
Shifts.	@		@	Shifts.
		Ore-house boss		
		Sorters........................		
		Trammers....................		
		Loaders........................		
		Loaders................		
		Screen-men..................		
		Samplers......................		
		Men on washers.....		
		Men on elevator............		
		Totals......................		
		Cars waste to dump.........		
		Cars loaded..................		

....

Ore-House Foreman.

* Size of original sheet, 8.5 by 5.5 in.

FORM. 7.—*The Portland Gold Mining Co.—Surface Report.**

Portland, Colo.,..............................190...

Shifts.	@	
		Men on dump-tracks.
		Men on cribbing......................................
		Men unloading lumber.
		Men unloading lagging and stulls.
		Men unloading cribbing timbers.
		Men unloading coal.
		Men unloading..
		Teamster and team..................................
		Painter..
		Watchmen.
		Men..
		Men..
		Men..

....
Surface Foreman.

* Size of original sheet. 8.5 by 5.5 in.

FORM 8.—*The Portland Gold Mining Co.—Carpenter-Shop Report.**

Portland, Colo.,..............................190...

Shifts.	@	Occupation.
		Carpenters in shop..................................
		Sawyers in shop......................................
		Sawyer helpers in shop.............................
		Carpenters on..
		Carpenters on..
		Carpenters on..
		Carpenters on..
		Carpenters on..
		Carpenters on..
		Men cutting lagging.................................
		Men on cribbing at...................................
		Men on cribbing at...................................
		....
		Foreman..
		Total..

....
Carpenter Foreman.

* Size of original sheet, 8.5 by 5.5 in.

FORM 9.—*The Portland Gold Mining Co.—Distribution of Labor in Machinery Department* (Size of Original Sheet, 11 by 6 in.).

	No. 1 Shaft.	No. 2 Shaft.	No. 3 Shaft.	Total.
Machinery, Repairs and Construction. *No...*				
On air-drill repairs........................				
Machinists on repairs...........				
Machinists on new work................				
Machinists' helpers........................				
Boilermakers.................................				
Boilermakers' helpers....................				
Pipe- and chain-gang......................				
Electrician.........				
Apprentice....................................				
Men on Engines and Compressors. *No...*				
Hoisting-engine, top.......................				
Hoisting-engine, underground.........				
Engine-wiper.........				
Compressor............				
Waste-haulage...............................				
Men on Boilers. *No...*				
Firemen..				
Boiler-cleaner...............................				
Pump-Men. *No...*				
Mine-pumps, station......................				
Mine-pumps, sinking......................				
Gulch-pump..................................				
Head pumpmen.............................				
Blacksmith-Shop. *No...*				
Blacksmiths..................................				
Blacksmiths' helpers......................				
Special Help. *No...*				
Haulage-system.............................				
Cables......................				
Cleaning, sorting, etc.....................				
....				
....				
....				
....				
....				
Total men......................................				

FORM 10.—*The Portland Gold Mining Co.—Blacksmith-Shop Report.**

Portland, Colo.,............. ....190...

Hours	No.	
		Blacksmiths sharpening drills.
		Helpers sharpening drills.
		Blacksmiths repairing drills.
		Helpers repairing drills.
		Blacksmiths making new drills.
		Helpers making new drills.
		Blacksmiths on general work.
		Helpers on general work.
		Blacksmiths on..
		Helpers on...
		Blacksmiths on..
		Helpers on...
		Blacksmiths on..
		Helpers on...
		Blacksmiths on..
		Helpers on...

....

Blacksmith Foreman.

* Size of original sheet, 8.5 by 5.5 in.

FORM 11.—*The Portland Gold Mining Co.—Mechanical Department.**

	Hours.	Total.
No.............. Hoist...............................		
No.............. Compressor.........................		
No.............. Boiler-room		
Dynamo-engine....................................		
Electric-lines..		
Waste-haulage.....................................		
Air-drills..		
Air-lines		
Steam-lines..		
Water-lines...		
Tram-cars...		
....		
....		
....		

Date...

Name...

* Size of original sheet, 5.5 by 4.5 in.

FORM 12.—*The Portland Gold Mining Co.—Order on Storekeeper.*

....190...

Storekeeper will please deliver to bearer and charge to..................................

....	.... (Size of original sheet, 4 by 6 in.)	....
....	....	....
....	....	....

....Foreman.

FORM 13.—*The Portland Gold Mining Co.—Ammunition Order.*

.................O'Clock Shift. ....190...

.................Level. Place Working ....

....................Sticks $1\frac{1}{4}''$ Powder. (Size of original sheet, 3.5 by 6 in.)

....................Sticks $\frac{7}{8}''$ "

............................Lengths Fuse ....Feet Long.

.... " " .... " "

.... " " .... " "

....Machine Man.

In sending in above order, Machine-Men must fill out the blanks indicating the level and place working. Always sign your orders.

FORM 15.—*The Portland Gold Mining Co.—Timberman's Report.*

............O'Clock Shift...............Date.

............Level...........Stope or Drift.

Posts,	
Caps,	
Sills,	
Ties,	
But caps,	(Size of original sheet, 6 by 4 in.)
Stulls,	
Lagging. New,	
Lagging. Raised,	
Sprags,	
Ladders,	
Planks, ft.,	

....

Timberman.

FORM 18.—*The Portland Gold Mining Co.—Trammer's Tag.**

.........O'Clock Shift,................190...

Ore.	Mill Dirt.	Waste.
....	....	....

Trammed from...............................

Level No........ and Run to..............

....

Taken from

Chute.	Platt.	Rough Bottom.
....	....	....

To be delivered in bin No................

....

Trammer.

Trammers will indicate whether "Ore," "Mill Dirt" or "Waste," by putting an X underneath the word; the same with "Chute," "Platt" or "Rough Bottom." Fill in and date blanks, sign name and send out with car.

If on other work than tramming, state number of hours.

* Size of original sheet, 5.5 by 3.5 in.

FORM 14.—*Portland Gold Mining Co.—Powder-Man's Report.**

Day.	Level.			Level.			Level.			Level.			Level.			Level.			Level.		
	Name of Place.			Name of Place.			Name of Place.			Name of Place.			Name of Place.			Name of Place.			Name of Place.		
	Powder.	Fuse.	Caps.	Powder.	Fuse.	Caps.	Powder.	Fuse.	Caps.	Powder.	Fuse.	Caps.	Powder.	Fuse.	Caps.	Powder.	Fuse.	Caps.	Powder.	Fuse.	Caps.
1......																					
2......																					
3......																					
4......																					
5......																					
6																					
7......																					
8......																					
9......																					
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23......																					
24......																					
25......																					
26......																					
27......																					
28......																					
29......																					
30......																					
31......																					

* Size of original sheet, 15 by 12 in.

FORM 16.—*The Portland Gold Mining Co.—Timber Report.**

Month of........................190...

	Name of Place.										Name of Place.									
Day.	Posts.	Caps.	Sills.	Ties.	Butt Caps	Stulls	Lagging.	Planks—Ft.	Sprags.	Ladders	Posts.	Caps.	Sills	Ties.	Butt Caps	Stulls.	Lagging.	Planks—Ft.	Sprags.	Ladders.
1......																				
2......																				
3......																				
4......																				
5......																				
6......																				
7......																				
8....																				
9......																				
10.....																				
11.. ...																				
12......																				
13......																				
14......																				
15......																				
16......																				
17......																				
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19......																				
20......																				
21......																				
22......																				
23......																				
24......																				
25......																				
26......																				
27......																				
28......																				
29......																				
30......																				
31......																				
Total..																				
Cost....																				

* Size of original sheet, 14 by 14 in.

Form 17.—*The Portland Gold Mining Co.**

Shift-Bosses' Distribution-Sheet. Month of......... ....190...

Level .... Place..

	Shift.	Machine-Men.	Machine Helpers.	Hand-Miners	Shovelers.	Trammers.	Pipe- and Track-Men	Timber-Men	Timber Helpers	Total Shifts	Cars Ore Hoisted	Cars Waste Hoisted.	Cars Waste Dumped in Stopes	Total.
1	8													
	6													
2	8													
	6													
3	8													
	6													
4	8													
	6													
5	8													
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	6													
28	8													
	6													
29	8													
	6													
30	8													
	6													
31	8													
	6													
Total...														

Labor-cost			All other expense.........		
Explosives			Total cost....................		
Lumber and timber			Tons ore produced........		
Machine-drills................			Per cent. shipped (est.)..		
Bosses, etc			Average assays.............		
Hoisting.........................			Estimated total value...		

* Size of original sheet, 14 by 8 in.

A Reference-Scheme for Mine-Workings.

BY WILBUR E. SANDERS, HELENA, MONT.

(Bethlehem Meeting, February, 1906.)

AT some period during the operation of metalliferous and other commercially valuable mineral-deposits in connection with their underground mining, when the developments therein have become so extensive that their description is tedious and confusing, some scheme for naming or numbering the various workings and their parts is necessary for convenience of reference. A simple and symmetrical yet expansive system of classification must be devised, one that is capable of being extended to cover all possible exigencies and conditions of future operations within the property.

Such a scheme, properly arranged, greatly simplifies communication between the various departments of the mining business, as, for instance, between the administrative and the operating departments of the concern, and is of the greatest assistance to the officers of the mine in enabling them to locate and describe particular points and parts of workings and specific kinds of work. Again, in the proper segregation and differentiation of mine-accounts, when they are carried out with that detail which will furnish ready information concerning the cost of a particular piece of work, or the various items and totals of expense connected with the development, operation and repair of each or all of the mine-workings, some such method of designation becomes a necessity. Moreover, such a plan is admirably adapted to the needs of the mining engineer, who, in connection with his reports upon mining-properties, is sometimes required, in his descriptions of certain blocks of ore, to locate lists of assays of samples taken from many and various places, and to designate workings and parts of workings that are located in different parts of the mine, sometimes at points wide apart.

To write out in full a sufficient description of any particular

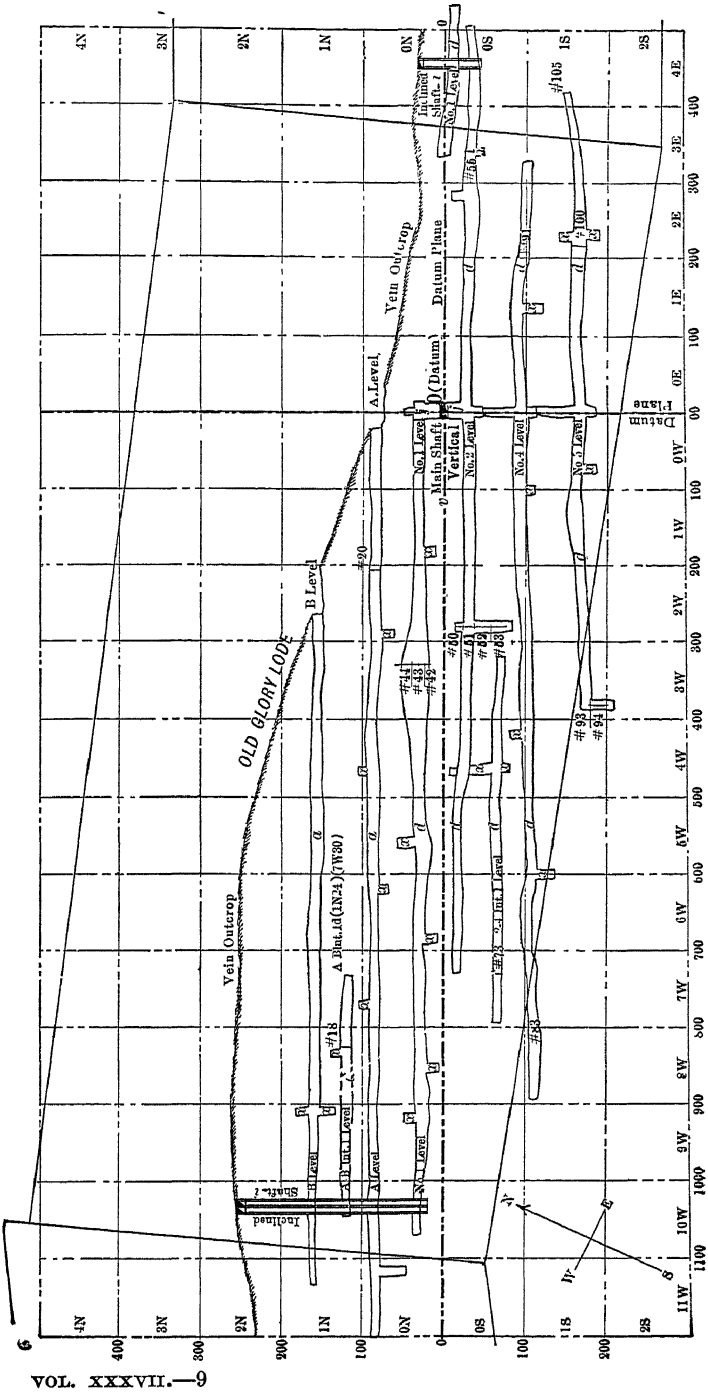


FIG. 1.—OLD GLORY QUARTZ-LODE MINING CLAIM, PLAN.

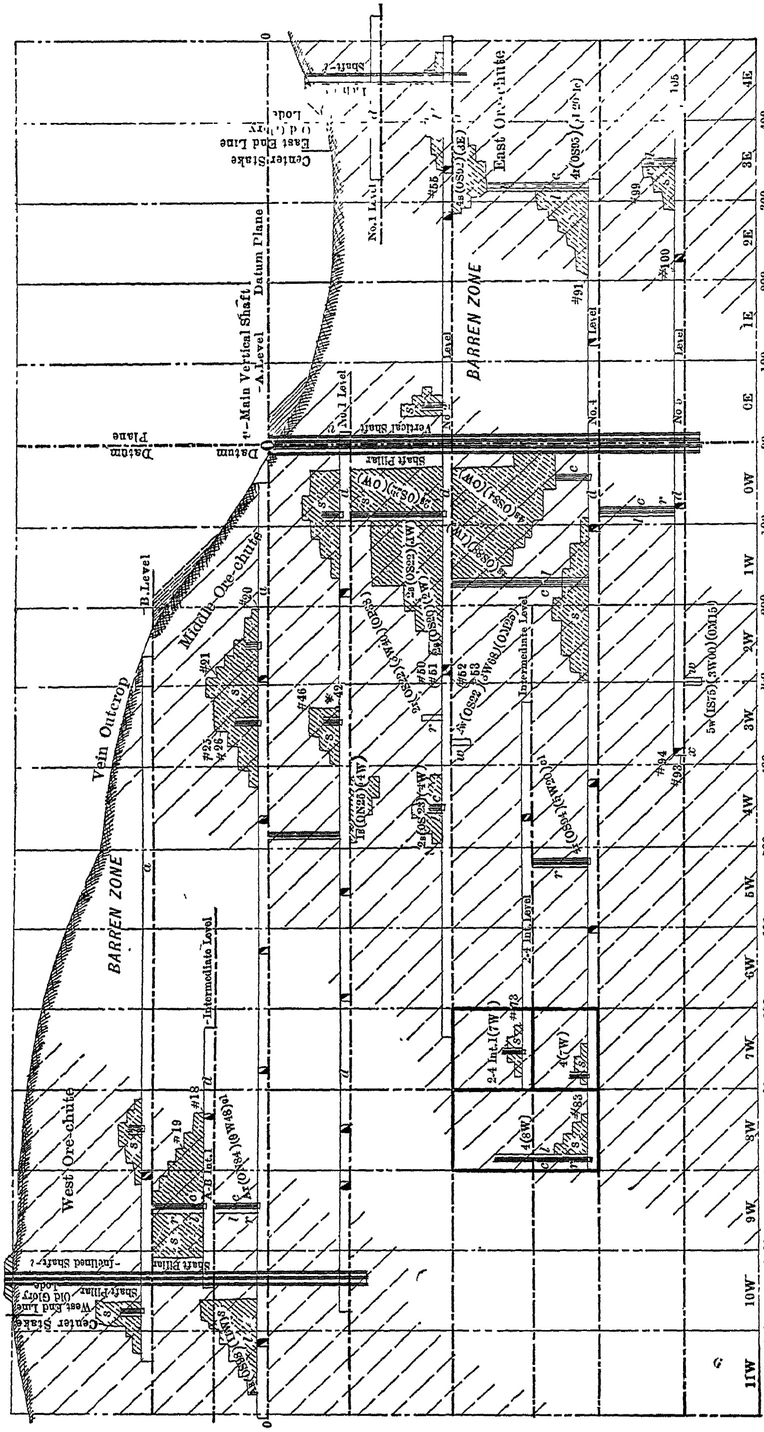


FIG. 2.—OLD GLORY QUARTZ-LODE MINING CLAIM. LONGITUDINAL SECTION.

locality or working of a mine, or even to explain the locations from which a lot of samples have been taken, would be far too cumbersome for practical purposes, and such a description would lack the essentials of systematic classification. For cases in which numerous and extensive developments are involved, the scheme may be founded upon some progressive system that will designate points and parts of workings and, by inference, the operations that are being carried on therein; and, these results being obtained, they should be capable of concise expression, stenographically, by symbols. The plan must be such in principle that it may be made to apply to all classes of underground mining, and be capable of fulfilling all the requirements that future extension of the mine-workings shall demand of it.

For the purpose of illustration, assume that the Old Glory quartz-lode mining-claim (Fig. 1, plan, and Fig. 2, longitudinal section), contains a single fissure-vein that outcrops upwards along the face of a hill, and which is developed within the vein along its strike, by the adits, *a*, of A and B tunnel-levels, and the interior drifts, *d*, of Nos. 1, 2, 4 and 5 levels, together with intermediate drifts and workings; the dip of the vein being at a steep angle towards the south. Furthermore, the mine is developed in depth by a main vertical working-shaft, *v*, and two subsidiary inclined shafts, *i*, the latter sunk on the vein. The levels of the mine comprise the adits, *a*, the storage-junctions or stations, *j*, at the junctions of horizontal workings with the principal inclined and vertical shafts, the cross-cuts, *x*, run from the stations of the vertical shafts to intersect the vein at the level of the different stations, the drifts, *d*, run from such cross-cuts horizontally along or near the vein, together with other connected and related interior workings.

Within the vein, for purposes of exploitation, ventilation and communication, and for the extraction of ores, other workings have been constructed; raises, *r*, have been carried upward from the several drifts and adits into blocks of ground overhead that contain the ore-deposits of the levels (when such bodies are attacked by overhand stoping), or into and through barren ground for purposes of ventilation; winzes, *w*, have been sunk below the drifts into blocks of ground that contain the ores belonging to the level from which the winze is sunk

when the deposit is attacked by underhand stoping, or that belong to the level beneath whenever they are extracted from that level; stopes, *s*, have been opened up for the purpose of winning the valuable materials from their places of deposit; and chutes, *c*, have been built up in order to bring the broken ores from the stopes to the drifts, that they may be drawn into vehicles and carried thence to surface, while cross-cuts, *x*, have been driven from the various drifts into the country-rock to either side of the vein for the purpose of prospecting its hanging-wall and foot-wall for ore-bodies, or evidence of valuable minerals. These are the workings that are of general utility in carrying out the processes of development and operation of a mining-property.

Since the mine-maps, drawn to a desirable scale, are a sufficiently exact reproduction of the workings of a mine, it follows that a plan for naming and numbering such workings for reference will begin with and be developed from the maps, and, for the sake of reference, such a scheme must be exhibited upon them. For the sake of brevity in description, certain symbols, letters or figures, are employed to designate the various mine workings, as follows:—

PASSAGEWAYS.

Horizontals:

a. Main and subsidiary exterior horizontal passageways, as adits and tunnels.

d. Main and subsidiary interior horizontal longitudinal passageways, as drifts, driveways, gangways, etc.

j. Storage-junctions, stations or platts.

x. Main and subsidiary interior horizontal transverse passageways, as cross-cuts, cross-headings, etc.

Inclines:

i. Main and subsidiary inclined shafts.

Verticals:

v. Main and subsidiary vertical shafts.

Ways:

c. Chutes, rock-ways, mill-holes, etc.

l. Ladder-ways, man-ways, etc.

r. Raises.

w. Winzes.

EXTRACTION PLACES.

s. Stopes, stalls, rooms, chambers, etc.
b. Backs, breasts, bottoms or sides, *in ore.*
p. Pillars of all kinds.

It is not probable that the course or strike of a vein will lie exactly in a N–S. or in an E-W. direction, but in a general way its strike and dip must be taken into account, and made use of for the purpose of locating the relative positions of the mine-workings, or any part or parcel thereof, with reference to some initial point of the mine, at which, properly, our scheme may be made to begin. The vein, therefore, lies horizontally in an easterly and westerly or northerly and southerly direction, and any particular working or part of a working is located and arbitrarily described as being east or west and north or south of our assumed initial point of beginning, the datum-point or 0 of the scheme. Direction with reference to this 0 point is symbolized by the initial letter of the bearing in capital form, thus: N. = North or northerly; S. = South or southerly; E. = East or easterly, and W. = West or westerly.

Similarly, as regards the elevations of mine-workings, or of parts or parcels thereof, being their vertical measurements above or below an assumed or accepted initial point, such distances with regard to that point are symbolized thus: P = (plus) above; M. = (minus) below.

Any and all points of mine-workings are referred, for purposes of location, to a system of base- or datum-planes "00" which are passed through the datum or "0" point of the scheme in three directions that are at right-angles to each other, as follows: a vertical transverse datum-plane, 00, passing through 0-point transversely across or at right angles to the course or strike of the vein; a vertical longitudinal datum-plane, 00, passing through 0-point parallel to the strike of the vein; a horizontal datum-plane, 00, passed through 0-point at right angles, or approximately at right angles, to the vertical datum-planes of the system. By means of these three planes any point whatsoever, be it north, south, east or west and above or below 0 of the scheme, may be referred to that datum (0) by means of co-ordinate measurements.

For purposes of measurement and description, three series of planes that are parallel to the above datum-planes are passed

through, in connection with the workings of the mine. Those planes, which are passed parallel to the vertical transverse and vertical longitudiual datum-planes, are established at intervals of 100 ft., while those which are passed parallel with the horizontal datum-plane are made to coincide with the planes of the floors of the different levels of the mine, whatever may be the vertical interval between the workings. By means of the datum-planes explained above, and the three series of parallel planes related to them and connected therewith, the rock-masses, surrounding and enclosing the workings of the mine, are divided and separated into cubical blocks, any point or part of which may be located with reference to the enclosing planes, or to 0 by means of longitudinal, transverse and vertical co-ordinates.

The series of vertical transverse planes, running north and south, are numbered in hundreds of feet, the distances between the different planes and the datum-plane being thus symbolized, as being 100, 200 *et seq.* to the east and to the west of datum-plane. Similarly, the series of vertical longitudinal planes running east and west are numbered in hundreds of feet to represent the distances north and south from the datum-plane of the different planes of the series, thus 100, 200 *et seq.* Since these planes are numbered in hundreds of feet, those blocks of ground immediately adjoining datum or 0-point of the system would be 0 hundred east, or west, or north, or south; for the reason that any point in such blocks would be 0 hundred east or in other directions plus the distance of that point from the 0-point; as a point 25 ft. north of 0 would be 025, preferably to be written 0N25, or similarly 0E25 if 25 ft. eastward from 0. Zero- or datum-block being thus established in each direction with reference to the scheme and to the 0-point, other blocks follow in sequence, as 1N, 2N, etc., 1E, 2E, etc., 1S, 2S, etc., and 1W, 2W, etc. By way of illustration, points 25 ft. from the beginning of certain blocks would read thus: 1N25 in block 1N; 4E25 in block 4E; 7S25 in block 7S; 12W25 in block 12W, the last of the above readings signifying that the assumed point is 1,200 plus 25 ft. west of 0 by the scheme. By thus describing and writing the location of a point, its direction or bearing, north, south, east or west, or above or below datum or 0, is so placed that it will separate the name or

number of the block in which the point is established, from the distance of that point within the block from the beginning thereof, as signified by the final numerals written as tens, thus 25, or 05, etc.

The series of horizontal planes which are made to coincide with the floor-planes of the different levels are symbolized by the designation of the levels themselves. The horizontal datum-plane is made to coincide with the floor of the main working-tunnel, or with the collar of the main working-shaft of the mine, whether the same is a vertical or inclined shaft, this being doubly the fact when, as in the Old Glory lode assumed, the top of that shaft coincides with the floor-plane of the main-working, A tunnel. Adit-tunnel levels, employed for the development of the mine above the horizontal datum-plane, are designated by the first letters of the alphabet in capital form, as A, B, etc., the floor-plane of "A" level being made to coincide with the datum-plane. The interior levels employed for the development and operation of the mine from the horizontal datum-plane downward to the deep are designated by the numerals 1, 2, 3 *et seq.* in descending order. Abnormal intervals between levels would call for an omission from the sequence in order that intermediate levels to be constructed in the future may properly be designated in the scheme.

Points within the blocks belonging to the various levels are referred to the planes of those levels, such points being usually above, though they may be below, the floor-planes thereof. For illustration, assume that the various points are established 25 ft. above (0P25) the floors of certain levels, within the stopes belonging thereto, which, under the scheme, would be designated as within the stopes of, and above, No. 1 level, 1s(0P25); as within the stopes of and above No. 5 level, 5s(0P25); as within the stopes of and above B tunnel-level, Bs(0P25). Similarly, a point assumed to be 122 ft. above (1P22) the floor of a level, as within the stopes of "B" tunnel-level, would be designated Bs(1P22). Likewise, a point assumed to be 33 ft. below (0M33) the floor of a level, as No. 1 level, would be designated 1s(0M33).

In designating points for reference under the scheme the first symbol to be employed should be that of the level to which a point properly belongs, as A, B, etc., 1, 2, 3 *et seq.*

Following the symbol of the level comes the designation of the working within which the point is established, as *a*, *d*, *j*, *x*, *i*, *v*, *c*, *l*, *r*, *w*, *s*, etc., from the legend. And succeeding the above should be written the symbols of location by which the point is located in space with reference to the datum-planes of the system, and thereby to datum or 0-point. These symbols of location should invariably be written in order, the one fixing the location of the point in a north and south direction from 0-point being first noted, the one fixing the location of the point in an east and west direction from 0-point being next noted, while the one fixing the location of the point above or below the plane of the level to which it belongs should be noted last, each symbol of location being enclosed in brackets. In illustration, a point established 105 ft. north of the longitudinal datum-plane, 1,111 ft. east of the transverse datum-plane, and within the stopes 89 ft. above the floor-plane of A tunnel-level, would read, As(1N05) (11E11) (0P89). Similarly, a point 1,221 ft. south of the longitudinal datum-plane, 207 ft. west of the transverse datum-plane and within the stopes 25 ft. above the plane of No. 3 level, would be designated, 3s(12S21) (2W07) (0P25). If, instead of being in the stopes above or below the plane of the levels, the two points here assumed were within the adits and drift thereof, and therefore practically at the floor-planes of the levels, the latter designation might be discarded and the descriptions would then read: Aa(1N05) (11E11), and 3d(12S21) (2W07).

Fig. 3 exhibits in plan two horizontal or drift-workings, as driven within the vein along the floor-planes of Nos. 1 and 2 levels, and shows the stations, bearings and distances established by the mine-surveys, together with the method by which the points of intersection of the various planes of the system with the workings may be located and fixed therein. In this exhibit, datum is located at the collar of the vertical main working-shaft, through which point are passed the vertical transverse, the vertical longitudinal, and the horizontal datum-planes at right angles to each other, for purposes of establishing and describing the locations of points and parts of workings of the mine.

In Fig. 3 it may be noted that the drift of level No. 1 bends towards the north to such a degree that the plane, 100 ft. east of

the transverse datum-plane, instead of intersecting the working at a point along its course that is just 100 ft. from the datum-plane or survey-station No. 1, intersects it rather at a point that is by survey-courses 50 + 40 + 23 = 113 ft. east of survey-station No. 1. Therefore, in fixing this intersection at its proper *locus* within the working, we would measure eastward from survey-station 1E, No. 2, a distance of 23 ft., and the point thus established would mark the beginning of block 1E, and should be so designated both within the working and upon the map. Similarly, from the survey-courses, the point of intersection of the 200-ft. plane, the commencement of block 2E of the system, with No. 1 level, drift is 217 ft. east of survey-station No.

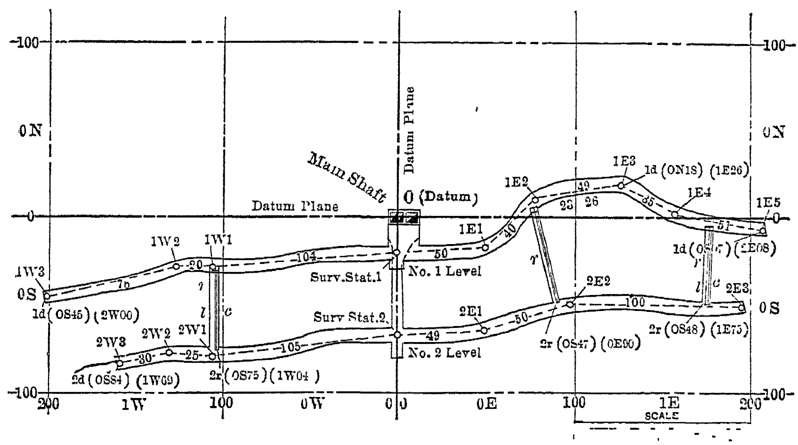


FIG. 3.—OLD GLORY QUARTZ-LODE MINING-CLAIM. PLAN OF TWO DRIFT-WORKINGS.

1, and 104 ft. east of the intersection of the 100-ft. plane with the drift. This point may be established and marked within the drift by measuring eastward a distance of 43 ft. from survey-station 1E, No. 4. The various points at which the divisional planes of the system intersect the mine-workings being established from the mine-surveys, and permanently marked within those workings, it then becomes easily possible to locate and describe any point within any particular block of ground at which any material exists or any particular piece of work is being carried on, both upon the map and within the mine.

The location of raises, man-ways, chutes and other similar interior workings may be established, and their designations marked within the main longitudinal workings; but, where

they are thus marked, certain symbols may be discarded for the sake of brevity, and only such as are essential to the description of the working be employed. Thus, the raise at point Ar(0N84) (9W48) of Fig. 2 could be marked within the adit as r(9W48), and, indeed, such markings may be still further abbreviated when fixed within the mine. If it is desired to specify that this raise is both a chute and a ladderway, then "c" and "l" of the legend may be added to its description. The designations of raises carried through from different levels at equal distances from the transverse datum-plane would exhibit the fact. Thus described and marked, there is no possibility of duplicating the number or designation of interior workings.

In the taking of a series of samples, the scheme may be made to describe not only the designation of each sample of the series and its width as well, but the exact location of the point from which they were taken. Samples taken from northward of the longitudinal datum-plane are numbered and measured towards the north, while those taken southward of that plane are numbered and measured towards the south. One or more samples taken from across the width of stopes above a particular point of a drift or other longitudinal passageway are, for convenience, referred to that point of the passageway with regard to the transverse and longitudinal designation thereof. The following list of samples assumed on the maps, Figs. 1 and 2, will sufficiently explain the workings of the system:—

Number of Sample.	Designation of Working.	Location.			Width of Sample.
		Transverse Planes.	Longitudinal Planes.	Horizontal Planes.	
18	A–B Int. 1d	(1N15)	(8W30)	....	5.
19	A–B Int. 1s	(1N15)	(8W70)	(0P40)	7.
20	Aa	(0N80)	(2W04)	....	8.
25	As	(0N80)	(3W55)	(0P60)	3.
26	As	(0N83)	(3W55)	(0P60)	5.
42	1d	(0N20)	(3W26)	....	6.
43	1d	(0N26)	(3W26)	....	7.
44	1d	(0N33)	(3W26)	....	4.
46	1s	(0N20)	(3W26)	(0P50)	12.
73	2–4 Int. 1d	(0S60)	(7W20)	....	5.
93	5d	(1S68)	(3W92)	....	5.
94	5x	(1S73)	(3W92)	....	15.
99	5s	(1S56)	(3E30)	(0P50)	6.

For example, the last sample would be written 99–5s(1S56) (3E30) (0P50)6.

The above reference-scheme for numbering and naming points and parts of mine-workings can be applied to the workings of any and all other classes of mineral-deposits, with such variations in its details as may be necessitated by changes in the methods of developing and operating such other deposits. In flatly-dipping veins the side of a drift or other longitudinal passageway that is toward the dip of the vein may be made the plane of division between the various levels of the mine and between the blocks of ore belonging to those several levels. In mass- or pocket-deposits of large extent and irregular occurrence, and in connection with bodies of ore that occur in the form of more or less irregular lenses, the plan above described may be of the greatest advantage in blocking-out the ores for purposes of description and localization; and it may be employed with great benefit in connection with the mapping of square-setted stopes. In the development of such bodies, and in operations connected with the winning of the valuable mineral products thereof from their places of deposit, for the sake of simplicity the various transverse and longitudinal horizontal passageways, or those which in the scheme under such conditions are arbitrarily assumed to be transverse or longitudinal workings, might well be made to coincide with the series of transverse and longitudinal planes which we have explained above.

The Geology and Petrography of the Goldfield Mining-District, Nevada.

BY JOHN B. HASTINGS, DENVER, COLO., AND CHARLES P. BERKEY, NEW YORK, N. Y.

(Bethlehem Meeting, February, 1906)

I. GEOLOGY.*

THE reconnaissance of the Goldfield mining-district, described in this paper, was made in May and June, 1905, and, though this time was too short for a complete report, the work accomplished may serve as a basis for more thorough future research. Dr. Berkey has examined the rocks collected by me, and I am using his classifications. As a result of further field-work, and Dr. Berkey's examination of the rocks, this paper is an elaboration and correction of my views expressed in the *Goldfield Sun*, May 12, 1905. The sketch-map, Fig. 1, illustrates the geology of the Goldfield mining-district.

Goldfield is an eruptive complex, consisting of alaskite (binary granite), hornblende-andesite, hornblende-dacite, rhyolite, pyroxene-andesite, pyroxene-dacite (sometimes containing a small quantity of olivine), quartz-felsite, olivine-pyroxene-andesite, and basalt. Considering this list to be arranged in the order of eruption, the age-sequence of the rocks conforms to Richthofen's law, formulated in 1868. From his observations in Europe and America, propylite, andesite, trachyte, rhyolite and basalt, occurring together, succeed one another. In a general way, the medium and the more acidic rocks precede the most basic—a result supposed by some to have arisen from their position beneath the surface, and due to a rude stratification by gravity in the original cooling magma during the older history of the earth, or to the same differentiation occurring during a quiescent (molten?) period preceding eruption, being aided by the ease with which acidic rocks, mixed with water vapors, become fusible. In effect, the more basic and less re-

* By John B. Hastings.

fractory rocks, from a smelting-standpoint, really seem to need a higher heat, or at least a more prolonged action, to bring them to the surface.

The observed occurrences of other rocks comprise: two inclusions of sedimentaries (indurated slates or shales, between alaskite and rhyolite) on Columbia and Vindicator mountains; a few very small areas of stratified tuffs, on the Tonopah Club and Desert Rose claims; a large body of tuff in Sections 4 and 33, and, possibly, in other areas of the altered zone, extending from Sections 3 to 36.

The ore-bodies occur in the hornblende-dacite and andesite of Knickerbocker mountain, in the rhyolite of Columbia and Vindicator mountains, and in pyroxene-andesite at the Quartzite mine. The whole area shows abundant remains of solfatarism, except in the rhyolite of Myers mountain, and the later pyroxene-andesite, olivine-andesite, pyroxene-dacite, felsite and basalt.

There is extreme silicification of the andesite, dacite and rhyolite along fractures generally northerly and southerly, but also easterly and westerly, forming reefs from 2 to 40 ft. wide, and, in the aggregate, possibly 40 miles long. These zones contain a vast quantity of material resembling hard sugar-quartz, but everywhere interspersed with a greater quantity of the original rock, still dimly showing its porphyritic character. Silicification, less intense, has occurred over large undefined areas.

The silicified reefs are usually barren, but the rich shipping-ores have been found intimately associated with them. My knowledge is insufficient to enable me to say much of the ore-deposits. In the Florence, Quartzite, and January mines, and in portions of the Combination mine, it appears as if the areas opened were on primary rich vein-fissures, which are rather small, except where they have been enriched.

In the Sandstorm mine, the rich ore follows a pre-existing northerly and southerly siliceous reef, along which the ore occurs at intersections of easterly and westerly fissures. While distinct brecciation was not observable at the crossings, in one instance the reef was faulted a distance of 6 ft. by the later fissure. The main ore-bodies were in the siliceous reef, the diminishing values extending for 25 ft. along the cross-fissures. The enclosing rock is rhyolite.

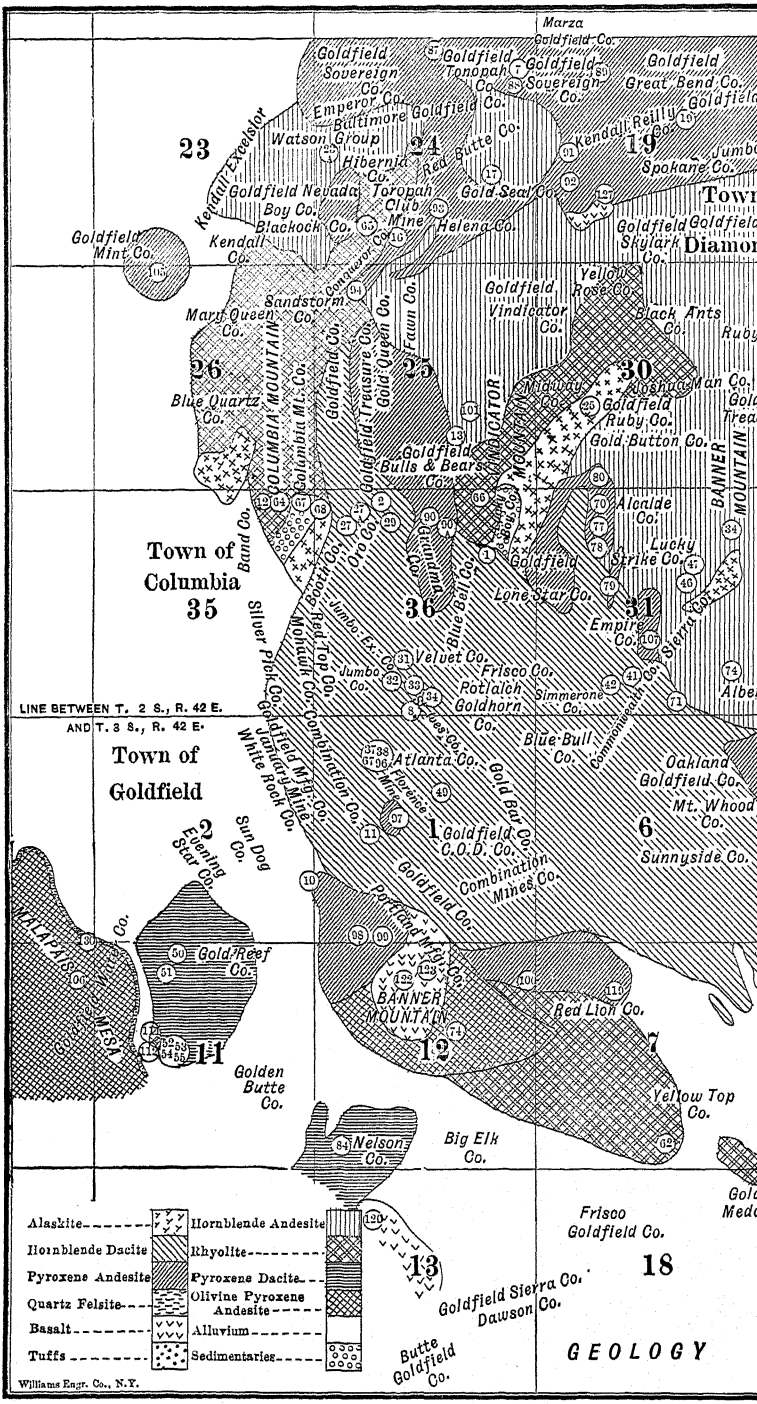
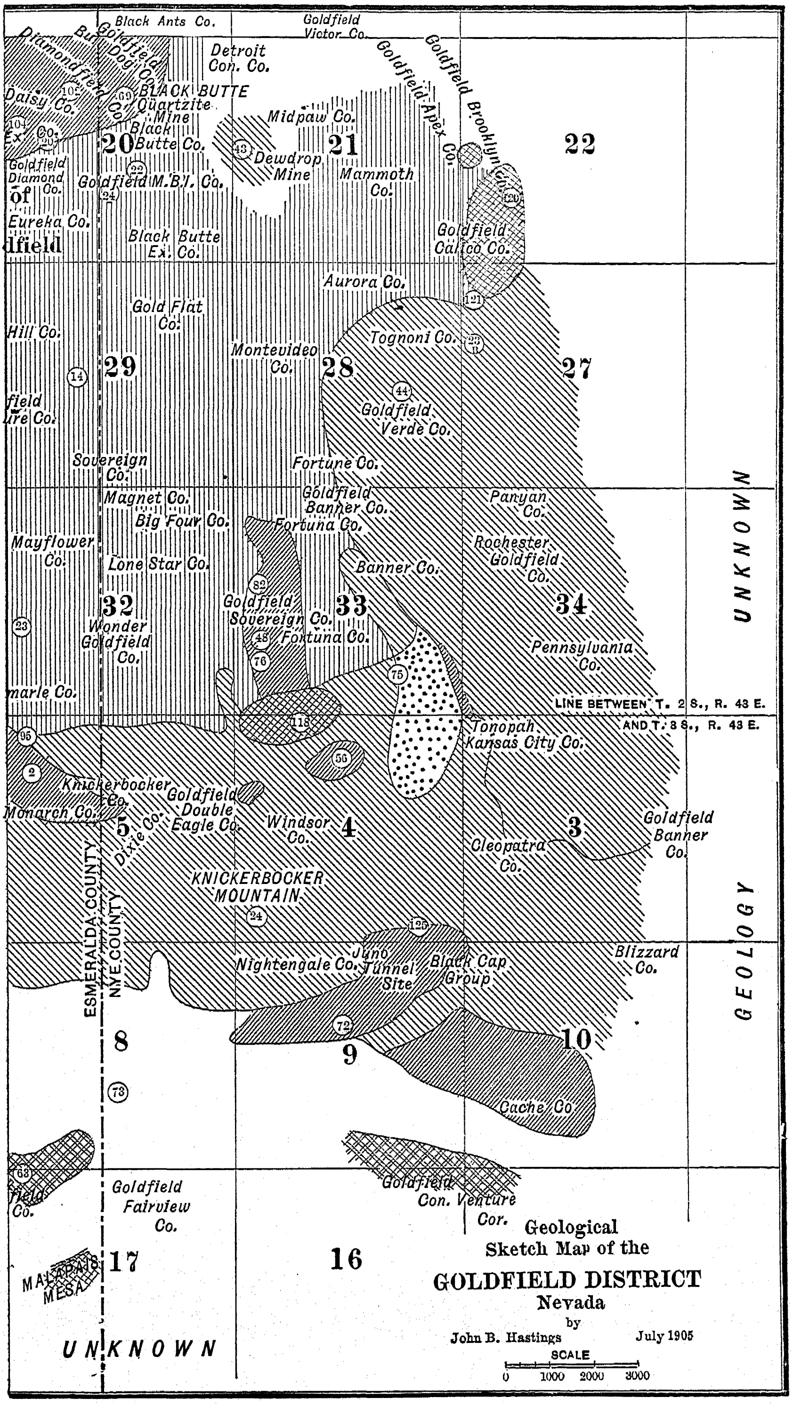


Fig. 1.—Geological Sketch-Map of



THE GOLDFIELD MINING-DISTRICT.

The Florence, Combination, and January veins, in the southerly area, striking NW. and dipping east, are crossed by other simultaneously mineralized fissures, striking NE. and dipping NW., into which the ore is deflected. The northwest fissuring continues, and the ore in the cross-fissures may follow them only for a certain extent, to be picked up again later in the parent fissures. The enclosing rock is dacite.

The Quartzite mine is the representative fissure in the northern end of the district. The vein strikes N. 20° W., dips westerly, and is about 5 ft. wide as stoped. A characteristic piece of the enclosing rock from the dump was pyroxene-andesite, having the same blue color and appearance as the southern mineral-bearing dacite, but lacked the quartzes. The brownish pyroxene-andesite, west of the quartzite, in Sections 19 and 24, is later than the silicified zones, peaks of the latter, too small to be mapped, outcropping like islands in the andesite-flow.

In the Florence, Combination and Quartzite, there are cross-fissures younger than the mineralization, which have faulted the veins, and this may be generally expected from the more recent vulcanism.

The tuffs on the Tonopah Club and Desert Rose, cursorily alluded to above, are in shallow detached bodies, occurring as shaly deposits on the present surface, and surviving only in a few depressions. At the Tonopah Club mine several of these small deposits, from 2 to 10 ft. deep, are crossed by the vein-fissure, with a slight faulting. The horizontal tuffs, with bedding-planes at right angles to the fissure, have been permeated and mineralized by the ascending solutions.

Since the beginning of the last century, attention has been called to the large amount of water and its vapor accompanying vulcanism; also to the fact that, after the cessation of rock-extrusion, hot waters still make a way to the surface, though forced from the body of the inactive eruptive to its contact with neighboring rocks and other lines of least resistance, such as faults, fractures and permeable limestones, etc. These waters, saline or vadose, were supposed to reach the volcanic rocks from the sea or from the earth's surface. It has been generally considered that, by gravity and capillarity, the waters would descend to the molten areas through saturated fissures and rocks, which, in turn, seemed to afford an easy passage to

the surface. On the other hand, some observers proclaim it impossible for water to travel against the pressure of vapors which would be created in rocks surrounding the deep scene of volcanic action.

It has been pointed out that but little capillary or other water is found in deep mines. In discussions on this subject I do not remember to have seen mentioned that fissures, later than the veins, now carry the surface-water. It is usual, in mining, to find the veins dry, and some cross-fissures flooded. Mine after mine has been cited as dry in its lower levels, and even the Rand mines are said to be making very little new water. Apart from local unfitness, I should imagine the center of such a synclinal as the Rand to be an ideal spot for artesian water. However, the chief fact seems to remain in prominence, namely, that there is much less water in underground circulation than was formerly supposed.

The most recent theory predicates the great central mass of the earth, beginning 190 miles below the surface (equaling 0.975 of its whole bulk), to be a core of compressed gases, similar to the sun's central nucleus; only, in the case of the earth, which has four times the density of the sun, these gases, on account of their specific gravity and resistance to deformation, must be nearly as heavy and as rigid as steel. As an addendum to this premise, water of volcanoes is then presumptively evolved from these gases, and escapes during a slight relief from the pressure which accompanies igneous phenomena.

Whatever may be the origin of volcanic waters and associated vapors and gases, their passage upwards, under great heat and pressure, thoroughly permeating large bodies of molten eruptives, would furnish the most ideal solvent in the best source of supply imaginable.

There is no proper drainage-system in Nevada. None of the rivers reach the ocean; all sink, either in the channel itself or in the lakes of the great valleys. Probably an important migration of vadose water progresses underground, which, in the past, has been interrupted by newly-formed volcanic necks. These waters became heated, joined magmatic waters, and rose to the surface; hence, as the rocks cooled and magmatic waters ceased, the springs dried up. Since there has been a succession of eruptive actions in Goldfield, there has also been one of

solfataric processes. Perhaps more than ordinarily, the Goldfield deposits suggest the derivation of the ores from ascending hot magmatic and vadose waters.

II. Topography.*

The Goldfield district has the erratic topography of a recent volcanic area, modified but slightly by erosion, the silicified portions forming small peaks by their resistance to the elements.

Columbia mountain, while composed largely on its south end of alaskite, as shown by the Columbia Mountain tunnel, is a rhyolitic neck; Vindicator mountain also is a rhyolitic neck.

Banner mountain, which may also be a volcanic peak, is possibly the product of erosion, since the summit is a mass of hard silicified andesite, and the surrounding basal plain a soft andesite and dacite.

Knickerbocker mountain is a great volcanic cone, from which the dacites have flowed northwesterly and easterly, partly covering the older hornblende-andesite.

Table mountain in Section 9, on which the Black Cap group and the Juno tunnel-site are located, is a basaltic neck, with a later flow of pyroxene-andesite, the hard andesite protecting the soft, underlying, altered dacite, which forms a ring around the protruding cone.

In Sections 12, 13 and 14 is another table-mountain protected by dacite and basalt, the two effusions exposed side by side.

Myers mountain, another volcanic neck, at first extruded rhyolite, in Section 7. This rhyolite is more basic than rhyolites of Columbia and Vindicator mountains, and macroscopically different from them. Olivine-pyroxene-andesite (Section 12) followed the rhyolite, then pyroxene-andesite (Section 1), and, finally, basalt (Section 12), now crowning the summit.

The Malapais mesa, west of Goldfield, extending 4 miles northerly and 4 miles southerly, and as much as 4 miles wide, is a marked example of a table-mountain protected by a hard cap. Here lake-beds and tuffs were covered with a thin bed

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* By John B. Hastings.

(from 10 to 50 ft.) of quartz-felsite; the felsite itself was covered soon after by a similar sheet of olivine-pyroxene-andesite. The pink felsite and black andesite are in marked contrast; the former is a fine example of igneo-aqueous fusion, with steam vesicles up to a diameter of 5 ft.; it may have been an underwater flow. The andesite is comparatively solid. These rocks seem like rivals, and one is struck with the tenacity with which the darker basic rock has so completely covered the lighter acidic rock. The depths of the flows differ, but not much, one from another; the edges are exposed for many miles on top of the bluff.

A somewhat interesting duplication occurs in Section 11, on the south end of the Wild Horse claim, which, at first, seemed suggestive of a downward faulting on the mesa-scarp. This

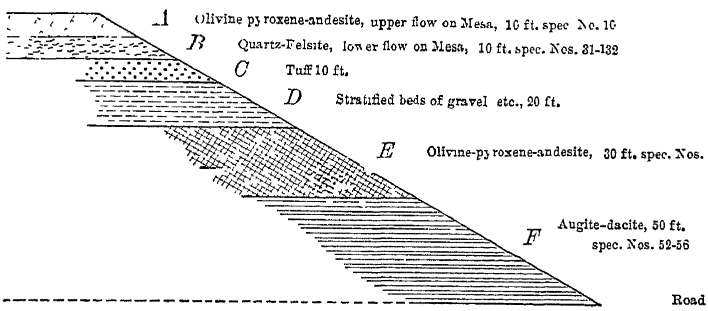

FIG. 2.—IDEALIZED GEOLOGICAL SECTION ON THE SOUTH END OF WHITE HORSE CLAIM, SEC. 11

occurrence is explained by Fig. 2. The augite-dacite, *F*, has a slight superficial resemblance to the felsite, *B*, and the olivine-pyroxene-andesite, *E*, to the olivine-pyroxene-andesite, *A*. This last rock preserves for miles its marked characteristics, perhaps the most noticeable being the iron-stained olivines. I could not determine whether *E* and *F* emanated southerly from under the mesa; they disappeared northerly under the wash. Perhaps the olivine-pyroxene-augite occurring in the wells of the Goldfield Brewery and Los Angeles Co. is connected with *E*. There is another nearby outcrop of olivine-pyroxene-andesite, a fine-grained rock, between the Water Tank butte and Mesa (Section 2). The dacite, *F*, is traceable in detached areas to and beyond the summit in Section 14, and east of the Lida road.

The mountains in Sections 27 and 34, on the east boundary of the district, are composed of a coarse typical andesite with quartz phenocrysts,—that is, hornblende-dacite much younger than the dacites of Knickerbocker mountain.

Mapped with the dacites of Knickerbocker mountain are fine-grained altered areas, the composition of which I was unable to determine.

III. Petrography.*

The chief object of the microscopic study of the specimens of rock, gathered by Mr. Hastings, has been to identify them and suggest relationships as an aid to field-mapping, and the interpretation of geological structure. At the same time, the characteristic secondary changes in the rocks, and the relationships of these to the ores, have been constantly in mind, with a view to their bearing upon the genesis of the ore-deposits of the district. Mr. Hastings, in the first part of this paper, has given the field-relationships, and makes the appropriate applications of such suggestions as a microscopic study of the rocks seems to warrant. On account of the interest felt in the district, and the rather unusual series of rocks represented at Goldfield, it seems advisable to add more detailed notes on the petrography of the area. The only notes published, so far as I know, are those of Mr. J. E. Spurr.[1] For the material and permission to use it in this way, I am indebted to Mr Hastings.

The specimens of rock and ore have been collected in a systematic study of the Goldfield district and vicinity. A greater part of these specimens necessarily represents the more obscure or less easily identified of the outcrops, together with the more complex structural areas. Altogether, however, the set must represent quite fully the range of rock-types of the district; and the comparative number of specimens in each type must indicate roughly their relative importance either areally, structurally, or economically.

The list of rock-types[2] includes,—(1), felsites, rhyolites, tra-

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* By Charles P. Berkey.

[1] *Bulletin No.* 260, *U. S. Geological Survey,* p. 133 (1905). *Bulletin No.* 225, *U. S. Geological Survey,* pp. 118–119 (1903).

[2] A similarity to the rocks of Tonopah, Nevada, is readily noted. The chief types described by Spurr for that area are: rhyolite, andesite, and dacite. *Bulletin No.* 219, *U. S. Geological Survey,* p. 12–15 (1903). *Professional Paper No.* 42, *U. S. Geological Survey,* p. 30–71 (1905).

chy-andesites, dacites, andesites, and basalts, representing felsitic and porphyritic varieties; (2), a granite as the only granitoid variety, except two or three cases of granular holocrystalline andesites and basalts that may, with equal propriety, be classified as diorite and dolerite, so far as this examination goes; (3), rhyolitic, dacitic, and andesitic tuffs among the pyroclastics; and (4), a carbonaceous quartz-slate as a representative of hydroclastics.

Andesites, the dominant type, include 78 specimens, or more than half of the total number. Next, numerically, is dacite, of which there are 21 specimens. Then follow the rhyolites to the number of eight specimens, and, finally, basalt, as which only three in the original identification list were classed, although there are five more, included with the andesites, that carry so much olivine and are of such strong basic affinities that they could, with equal reason, be included with the basalts. No other type occurs in large numbers of specimens.

Silicified, indurated rocks include the specimens of ores and several others, and are treated separately.

In the following list the numbers of the specimen correspond to the numbers on the map, Fig. 1, and are the serial numbers preserved with them and the corresponding thin sections in the petrographic laboratory of Columbia University, New York, N. Y., and by Mr. Hastings at Denver, Colo.

Granite (No. 1). The rock is very coarse-grained, with an unusually large amount of quartz, which carries numerous inclusions, largely in trains, some passing from one grain to another. Cracks also extend across in a similar manner, showing, on the whole, considerable crushing and recementing. The rock has no other gneissic characters. Orthoclase is abundant and uniformly clouded with kaolin-alteration, making it nearly opaque. There is a little albite. Iron oxide, hematite, gives a brown stain in a few spots. No identifiable dark mineral is seen in the slide. The rock is an exceptionally acid variety of granite, and is a fair representative of the alaskite of Spurr.

Rhyolite (Nos. 62 to 68). Most of these rocks show a characteristic flowage-structure, and a spherulitic or mottled, devitrified ground-mass. Quartz is abundant, both in the groundmass and in rounded or angular phenocrysts, sometimes giving

a decided porphyritic aspect. Resorption of the phenocrysts is marked. Biotite is the chief dark mineral. Orthoclase predominates over plagioclase. Magnetite is abundant in one or two specimens. The dark minerals are not conspicuous. Scapolite is developed to a considerable extent in three specimens, taking, in part, the place of original feldspars. A mottling of the rock, due to silicification, is also noticeable. No attempt has been made to indicate the field-relationships or the history by a differentiation into rhyolites and quartz-porphyries.

Felsite (Nos. 129, 130). There are two rocks of acid character, with glassy and devitrified ground-mass, showing flowage, that are called felsites, chiefly because of inability to decide whether rhyolitic or dacitic affinities prevail. Quartz is the only abundant determinable mineral, although both orthoclase and a plagioclase are present. Biotite and pyroxene are also present. The two rocks are not alike, however, and evidently do not belong to the same flow. They are regarded as a sub-type between the characteristic rhyolites of the district and the dacites proper, both of which are well recognized.

Dacite (Nos. 27 to 61). This group shows considerable variation. There are no less than eight sub-varieties or sub-groups, based upon minor specific differences among them, as seen in the microscope. One specimen, at least, has about an equal claim to classification with the rhyolites, and from this they represent many different mixtures and proportions to the other extreme, where one specimen of decidedly basic character, and in which olivine and augite are prominent constituents, can, with even better propriety, be placed with the quartz-basalts. The ground-mass of these rocks is either glassy to felsitic, or spherulitic and mottled, or pilitic. Flowage is a prevailing structure, specially marked in those of pilitic habit, and is well-developed even in the more crystalline varieties. Nearly all are porphyritic, although the phenocrysts are mostly very small, or, at best, inconspicuous, except as seen in thin section. Plagioclase is abundant as phenocrysts. The medium types all carry biotite in considerable amount as the chief dark constituent (Nos. 28, 29, 30). Hornblende is present with biotite, and in Nos. 31, 32, 41, 42 and 85 it becomes the chief ferro-magnesian constituent, and is developed in fine idiomorphic individuals. Most of those, low in biotite and hornblende, carry aug-

ite, showing the same range as is seen in the group of andesites proper, and the extreme in this direction is reached in the specimens before noted, in which olivine also appears. Therefore, as outlined above, the series, in which quartz is a prominent constituent, exhibits the complete range from rhyolite and quartz-felsite, through mica-dacites, hornblende-dacites, augite-dacites, to quartz-basalts.

Trachy-Andesite (3 specimens). There are no true trachytes. One specimen in particular (No. 94), and two others to less marked degree, are decidedly more acid than the average andesite, and have a trachitoid structure. The relative abundance of the different feldspars is in question. But the relative position of these rocks, among the others of the whole series, is best indicated by the above name.

Andesites (78 specimens, subdivided below). The andesitic rocks are in greatest number, surpassing all other types combined, and exhibiting the greatest number of minor differences among themselves, which indicate separable structural field-units of flows, sheets or dikes. The more basic varieties are the most perfectly preserved rocks of the district. The large number involved, and the evident importance of the group, make it advisable to subdivide into specific mineralogical varieties. Porphyritic development is prevalent, though often on a microscopic scale. Those most notable are referred to in the geology of the district as andesite-porphyries. In the present discussion, however, these textural variations are largely ignored in the effort to indicate the mineralogical relationships. They cover the entire range from trachy-andesites to the basalts proper.

(a) Mica-Andesite (3 specimens). Biotite is the chief ferromagnesian mineral. Hornblende is either negligible or is present in subordinate amount. Chlorite is a common secondary (Nos. 3, 4, 17). In several much altered specimens the ferromagnesian mineral is destroyed. Some of them probably belong in this group.

(b) Hornblende-Andesite (11 specimens). Hornblende is the chief dark mineral (Nos. 5, 19, 20, 26), occurring abundantly as phenocrysts with resorbed borders, and frequently attacked by chloritic alteration. Plagioclase is, however, the chief idiomorphic mineral, sometimes showing exceptional zonal growth

and giving a marked porphyritic texture (Nos. 20, 19). Biotite is usually present in minor amount. The ground-mass is mottled or exceedingly fine felsitic or pilitic in texture. Occasionally, there is excessive alteration and induration of the whole rock (No. 45), and the structure and appearance of the product is precisely the same as specimens of ore. Iron oxide and calcite are common secondary products. One rock included here is heavily impregnated with iron oxide in small grains or specks that are so persistent and uniformly distributed as to suggest simple oxidation of a pyrite-rock. It is otherwise also much modified in composition by the usual weathering products, so that nothing but these and a few structural outlines remain to indicate its relationships (No. 23). The group, in part, includes rocks that carry little dark mineral and are of acid character. A great number are of medium andesitic composition.

(c) Pyroxene-Andesite (augite-andesite), (41 specimens). By far the greater number of andesites carry pyroxene. Common augite is the prevailing species. There are 41 pieces of this class, exclusive of those carrying olivine also, which are discussed separately; 14 specimens have little pyroxene, or, indeed, any other ferro-magnesian constituent. A few of them are among the most acid of the andesite group. As a rule, they have a fine pilitic ground-mass; they usually show flowage, and a porphyritic texture prevails. Resorption of the phenocrysts and zonal growth are almost constant characters; 18 have a trachytoid structure, with chiefly lath plagioclase instead of orthoclase in the ground-mass; four (Nos. 88, 86, 83, 76) have biotite, in addition to augite, as a dark constituent. Hornblende is the additional basic mineral in eight pieces, though it is abundant in only two (Nos. 16 and 85), and there is an occasional grain in many others; four are strongly porphyritic, giving a strikingly granular aspect to the ground-mass, which, with higher magnification, exhibits multitudes of stout pyroxene rods. This variety is also the extreme in basicity of the simple pyroxene-andesite group (Nos. 72, 73, 74). Two specimens (Nos. 87 and 24) have a dense, stony, perhaps devitrified, ground-mass, giving a distinct structural variety. One (No. 98) has zonal growth of the feldspars and pyroxenes developed to an extraordinary degree—some of them also

marking resorption stages. One specimen (No. 12) is distinctly fragmental in its larger units, but is considered a flowage breccia rather than a tuff. As a group, the augite-andesites are of great variety in minor characters, and represent numerous different field-units.

(d) Hypersthene-Andesites. In three specimens hypersthene occurs as an essential constituent, in addition to augite and some hornblende (Nos. 92, 100, 105). They are not all of the same structural type.

(e) Olivine-Bearing Pyroxene-Andesite (20 specimens). This whole sub-group carries olivine in addition to augite. With rare exceptions, hornblende and biotite are absent. The set of specimens here classified ranges from four specimens showing very little olivine to an equal number at the other extreme, where olivine becomes the chief basic constituent, and the general basic composition so nearly balances the feldspathic component that they may, with equal propriety, be included with the basalts. In a large majority of these cases, however, the light-colored, leucocratic minerals are in excess of the melanocratics. The type is, therefore, of considerable abundance in the district. Labradorite is the abundant feldspar, but anorthite appears in the most basic. Three specimens show a tendency to diabasic structure. One (No. 116) is also rather coarse-grained for this group, and therefore has dolerite or diabase affinities (No. 125). These rocks are, for the most part, strongly porphyritic. They are not so prominently modified by alteration as many of the more acid groups. They are not indurated, and in a large number the olivines are still very fresh. Two of the specimens have a notably granular ground-mass (No. 113), due to pyroxene of the second generation. A fine pilitic or trachytoid ground-mass is more common. One (No. 127) shows marked flowage-structure and zonal growth of the phenocrysts to an unusual degree—16 stages appearing on one feldspar. All of the constituents occur as phenocrysts. Magnetite is very abundant in some cases.

Quartz-Basalt.—One of the most basic of the above series (No. 115) carries quartz phenocrysts as prominent constituents. Olivine is plentiful and augite is very abundant,—both together at least equalling the feldspars in the rock. The quartz occurs in large grains, showing resorption, and surrounded by a mass

of granules and columns of augite resembling the Lassen's Peak quartz-basalt, described by Diller. The rock forms the basic limit of the quartz-bearing porphyries of the district, already mentioned under the dacites.

Basalt (3 specimens).—All are olivine basalts of the same general character as the most basic described under the olivine-bearing pyroxene-andesites above. These are still more basic (Nos. 122, 123, 124). Olivine occurs in the ground-mass and as phenocrysts. Much of the augite is granular or micro-idiomorphic in the ground-mass. In relative abundance the constituents are augite, plagioclase, olivine, magnetite. The rocks, which form the basic end of the series at Goldfield, are fresh, the olivines showing staining only along the cracks.

Alteration-Varieties.—A few of the specimens show unusual or exceptionally complete alteration. No. 7 was originally an olivine-bearing augite-andesite. The traces of original structure are still so well preserved that the zonal growth of former feldspar-areas can be seen. But the rock is now largely composed of a very feebly porlarizing substance, so feeble in places as to require a sensitive tint to detect the change, and this has taken the place not only of the feldspar and olivine, but of any other constituent. It is regarded as chiefly serpentine. The rock is a pseudomorph throughout. Three of the rocks, and, to a lesser degree, a few others, have developed scapolite. They are, where determinable, types of dacitic affinities. Pyrite is abundant in them, and secondary quartz is present in large amount. Scapolite has formed at the expense of the feldspar. Traces of original biotite and hornblende occur, and magnetite is the only prominent original mineral left, except an occasional quartz phenocryst. A good many of the slides show silicification. The mottled quartz-bespattered ground-mass, so often seen in the more acid and more altered varieties, is doubtless due to this process. It is a common condition in these rocks, and is especially marked in the ores and in those specimens heavily charged with pyrite. Silicification is the most characteristic alteration-effect seen in the rocks of the district.

Volcanic Ash, Tuff.—Several specimens are extremely fine-grained, and lack all the structures that are so persistent in the other rocks. They are evidently much altered also, which adds to the uncertainty of their identification. The coarser ones are

more satisfactory, and are certainly tuffs. The finer ones are called volcanic ash (Nos. 133–137),

Shale, or Slate.—Two specimens (Nos. 140 and 141) are very uniformly granular, black, very fine-grained, in which quartz, sometimes interlocking, can be identified. Besides, there are multitudes of uniformly distributed black specks, in part, perhaps iron oxide, but, in part, thought to be carbonaceous matter. Small quartz-veinlets abound, especially in No. 140. The rock, a sedimentary one, is a silicified carbonaceous shale, and occurs as an inclusion in one of the trachy-andesites.

Ores.—Not more than ten of the specimens are ores. They are all quartzose. The interlocking-grains are small, prevailingly of microscopic size. In some specimens, all original structures are obliterated in the process of silicification and mineralization that they have endured. In a few, however, there are satisfactory traces of original structures and minerals,—such as original phenocrysts,—that prove the igneous character of the original rock. The forms of the phenocrysts have persisted after everything else has been lost. There is no doubt in these cases that the ores are essentially silicified porphyry, and that the mineralization has accompanied silicification. Scapolite occurs in some of these cases. A few show previous crushing, the traces of brecciation being still preserved. Therefore, the silicification probably follows these crush-zones through the formations. Hematite (No. 138) occasionally holds the place of original pyrite. But in the best slides four metallic minerals occur—all perfectly fresh,—gold, pyrite, a massive or granular reddish-bronze, dark, metallic mineral (unidentified), and a whitish-gray columnar or acicular metallic mineral, probably bismuthinite. There are strong signs of bismuth in fragments of the ore, and it is thought that both the last two minerals (No. 40) carry that element. The metallics are promiscuously distributed through the siliceous matrix, and, in the same manner, through and among each other. Doubtless, they were all formed in the process that involved the silicification of the rock, and, in these cases at least, they have suffered little change.

IV. List of Specimens.*

The following list of rock-specimens gives the specific localities and the identification; the numbers correspond to those on the map, Fig. 1.

1. S. 85 E., 250 ft. from NW. cor. Examiner, Section 36. Granite.
2. N. 20 E., 300 ft. from S. side center Huntch Bell, No. 18, Section 5. Felsitic andesite.
3. Half-way between Banner and Knickerbocker mountains, Section 32. Mica-andesite.
4. A piece from near the flagstaff on Banner mountain, as little altered as could be found, Section 31. Indurated mica-andesite.
5. Butte 250 ft. SW. of SW. cor. of Blind Ledge, Section 33. Hornblende-andesite.
6. Butte 1,200 ft. S. of ¼ cor. to Sections 4–33, Section 4. Andesite.
7. Fine-grained altered rock from shaft back of hill on S. end of Jumbo, Section 1. Andesite.
8. Float from St. Ives, probably came out of workings S. of shaft, Section 36. Altered andesite.
9. Dump about 800 ft. S. (?) of Portland Mining Co.'s shaft, Section 12. Olivine-andesite.
10. Butte adjoining W. side center Gold Coin, Section 2. Pyroxene-andesite.
11. Rustler Fraction W. side center, Section 1. Andesite.
12. Breccia from Piedmont Fraction ; this outcrops at foot of hill below the rhyolite, Section 35. Andesite flowage-breccia.
13. Gold Standard discovery, Section 25. Andesite.
14. Ridge at W. side cor. Gold Watch, freshest piece to be found, Section 29. Felsitic andesite, scapolite rock.
15. Little E. of N. side cor. Utah, Section 24. Andesite.
16. Tonapah Club, SE. cor., Section 24. Pyroxenic hornblende-andesite.
17. N. 10° W., 100 ft. from Belmont Queen discovery, Section 24. Mica-andesite.
18. Oakes discovery, Section 24. Andesite.
19. Half-way between Victor and Great Bend, Section 19. Andesite.
20. Between Diamondfield and the Black Butte claim, Section 20. Andesite.
21. Outcrop near Nye and Esmeralda Co.'s boundary stake, on road to Diamondfield, Section 20. Andesite.
22. 200 ft. S. of SW. cor. of Black Butte, Section 20. Andesite.
23. N. 65° E., 140 ft. from Calico discovery, Section 22, the freshest piece to be found in a large area. Andesite, badly weathered.
24. Dike on Willows claim, Section 27. Andesite with glassy ground-mass.
25. 100 ft. NE. from SE. cor. Hesperion, Section 30. Trachyte tuff, probably.
26. Knickerbocker mountain, exact location lost, Section 5. Indurated andesite.
27. Oro Goldfield shaft dump, Section 36. Dacite-porphyry.
28. Same, a little away from the shaft. Dacite-porphyry.
29. Same, from this vicinity, Section 36. Dacite-porphyry.
30. From low hill near NW. cor. of St. Louis. Dacite-porphyry.

* Collected by Mr. Hastings and determined by Dr. Berkey.

31, 32. Jumbo Ex. dump, Section 36. Dacite-porphyry.
33, 34. From open cut N. of St. Ives shaft, Section 36. Dacite-porphyry.
35. Knickerbocker mountain summit, Section 4. Dacite-porphyry, indurated.
36. Typical sample of the country-rock carrying the veins. Dacite-porphyry.
37. Florence rock from the dump, Section 1. Altered dacite, scapolite rock.
38. Ditto, heavily silicified, Section 1. Indurated dacite, ore.
39. Rich mill-rock, Florence. Indurated dacite, ore.
40. Florence ore.
41. About 300 ft. E. of Minnevada tunnel, Section 31. Dacite-porphyry.
42. Ditto, 200 ft. E. of tunnel. Dacite-porphpry with pyroxene.
43. Dewdrop shaft, Section 21. Dacite.
44. Gold Fountain, SE. cor., Section 20. Hornblende-dacite-porphyry
45. Huntch Bell, S. side center, Section 5. A much-altered rock, either andesite or dacite.
46. NE. cor. Midway Fraction, Section 31. Dacite.
47. Ditto, from N. end of flow. Dacite.
48. New York No. 1 discovery, Section 33. Silicified porphyry.
49. 500 ft. SW. of SW. cor. Union Jack, Section 1. Silicified and scapolitic rock.
50, 51. Butte, 1 mile S. of town, Section 11. Dacite-porphyry.
52. Wild Horse claim, S. end, lower flow butting against mesa, Section 11. Dacite.
53, 54. Wild Horse claim, S. end, lower flow butting against mesa, Section 11. Olivine-bearing pyroxene-dacite.
55, 56. Same as the last three, but east of the road. Dacite-porphyry.
57, 58. Regarded as the same rock as No. 52, from bluff east of Lida road, Section 14. Dacite.
59. Detached area S. of No. 59. Dacite.
60. Top part of No. 57 flow. Olivine-bearing pyroxene-dacite.
61. Valley View—Eclipse claims, from top of the Table-mountain on which these claims are situate, Section 12. Dacite.
62. Meda claim, Section 7. Spherulitic rhyolite-porphyry.
63. Cor. to Sections 7, 8, 17, 18, Section 17. Rhyolite.
64. West slope of Columbia mountain. Rhyolite.
65. SE. cor. Desert Rose, No. 1, the rock with abundant quartzes E. of Sandstorm, Section 24. Rhyolite.
66. Lucky Boy amended discovery ; the slates are 125 ft. wide, and west of them this rock outcrops, Section 36. Rhyolite.
67, 68. From top of Columbia mountain. Rhyolite.
69. Quartzite shaft dump, Section 20. Pyroxene-andesite.
70. Alhambra, S. side center, Section 31. Pyroxene-andesite.
71. Just E. of intersection of Commonwealth and N. Y. side line, Section 31. Pyroxene-andesite.
72. 600 ft. SE. of NE. cor. Ida May, Section 9. Pyroxene-andesite.
73. Grasshopper Nos. 1 and 2, 50 ft. E. of E. and W. side center, Section 8. Pyroxene-andesite.
74. 75 ft. W. from N. side center Huntch Bell, No. 9, Section 31. Pyroxene-andesite.
75. Chicago, SW. cor., Section 33. Pyroxene-andesite.
76. N. Y. No. 1 discovery shaft, Section 33. Pyroxene-andesite.
77, 78. True Bell, W. side center, Section 31. Pyroxene-andesite.
79. E. side center Lady June, Section 31. Pyroxene-andesite, chloritic.

80. 50 ft. E. of SW. cor. of Fraction claim at foot of east slope of Vindicator mountain, Section 30. Pyroxene-andesite.
81. NE. cor. Gold Standard, Section 25. Pyroxene-andesite.
82. N. Y. No. 2, W. side center, Section 33. Pyroxene-andesite.
83. S. 20° E. 200 ft. from SW. and SE. corners Elizabeth and Jack Rabbit Fraction, Section 31. Pyroxene-andesite.
84. Oakes discovery shaft, Section 24. Pyroxene-andesite.
85. 200 ft. SW. of SW. cor. Adams, Section 25. Pyroxene-andesite.
86. Blackrock discovery, Section 24. Pyroxene-andesite.
87. Sections 13–24. Pyroxene-andesite.
88. Cloudy Day dike? from shaft, Section 24. Pyroxene-andesite.
89. Red Butte, NE. cor., Section 19. Pyroxene-andesite.
90. From hummock east of low hill near NW. cor. of St. Louis, Section 36. Pyroxene-andesite.
91. N. 70° E. from SE. cor. Red Butte, Section 19. Pyroxene-andesite.
92. Athabasca, E. side cor., Section 19. Pyroxene (hypersthene)-andesite.
93. Autumn discovery, Section 24. Pyroxene-andesite.
94. Butte near Adams shaft, Section 25. Pyroxene-trachy-andesite.
95. S. 64° W., 100 ft. from NW. cor. Huntch Bell, No. 3, Section 5. Pyroxene-andesite.
96. Area close to Florence shaft on the east, Section 1. Pyroxene-andesite.
97. Rustler Fraction, N. 30° E. from E. side center, Section 1. Pyroxene-andesite.
98, 99. From hill west of Myers peak, Section 1. Pyroxene-andesite.
100. Red Lion boundary, Section 12. Hypersthene-andesite.
101. Location unknown. Pyroxene-andesite.
102. 100 ft. E. of S. side cor., Section 20. Pyroxene-andesite.
103. Top of bluff on summit, east of Lida road, Section 14. Pyroxene-andesite.
104. Summit of butte, south of Blue Bird, Daisy No. 2, and 3 Friends Fraction, and west of Black Butte, Section 20. Pyroxene-andesite.
105. Detached dome west of Tonopah road, Gold Leaf discovery, Section 26. Hypersthene-andesite.
106, 107, 108. Upper flow on mesa west and southwest of town, Section 11. Olivine-bearing pyroxene-andesite.
109. Top of butte where tanks of Goldfield Water Co. are situated, Section 2. Olivine-bearing pyroxene-andesite.
110. Upper flow on mesa at head of gulch at Wild Horse claim, A of Fig. 1, Section 11. Olivine-bearing pyroxene-andesite.
111, 112. E. of Fig. 1, south end Wild Horse claim, Section 11. Olivine-bearing pyroxene-andesite.
113. From behind butte on which water-tank is situated, at mouth of gulch west of town. Dike? Section 10. Olivine-bearing pyroxene-andesite.
114. Goldfield Brewery well, Section 3. Olivine-bearing pyroxene-andesite.
115. Well at Los Angeles Co.'s mill west of town, Section 2. Quartz-basalt.
116. Fifty-foot well, S. 10 E., 660 ft. from Goldfield Hospital, Section 2. Olivine-bearing pyroxene-andesite.
117. Area, Section 12. Olivine-bearing pyroxene-andesite.
118. 1,650 ft. E. of cor. to Sections 4, 5, 32, 33. Olivine-bearing pyroxene-andesite.
119. NW. cor. Red Lion Fraction, Section 7. Olivine-bearing pyroxene-andesite.

120. Capping of hill, 100 ft. E. of W. side center Anaconda, Section 22. Olivine-bearing pyroxene-andesite.
121. Willows No. 1, near W. side center, Section 27. Olivine-bearing pyroxene-andesite.
122, 123. Myers Peak, Section 12. Olivine-basalt.
124. From top of bluff, a little SE. of summit, on Lida road, Section 14. Olivine-basalt.
125. Alvarado, SE. cor., Section 13. Olivine-bearing pyroxene-andesite, near basalt.
126. Grubstake Fraction, on S. side of SW. cor., Section 4. Olivine-bearing pyroxene-andesite, near basalt.
127. Highwater, SW. cor., Section 19. Olivine-bearing pyroxene-andesite, near basalt.
128. From top of bluff, a little east of Lida road, after crossing summit, Section 14. Pyroxene-andesite.
129. Lower flow on Malapais mesa, near divide on Lida road, S. of town, W. of road, SE. cor. of map. Felsite.
130. Normal piece from lower flow on mesa, SW. of town, Section 11. Felsite.
133, 134. Desert Rose No. 1, Section 24. Volcanic ash.
135. Tonopah Club, Section 24. Volcanic ash.
136. Locality unknown. Volcanic ash.
137. Near Tinhorn, Section 5. Volcanic tuff.
138. An indurated rock of doubtful identity.
139. West slope Columbia mountain, near foot, Section 35. Tuff.
140. Lucky Boy, inclusion in alaskite and rhyolite, Vindicator mountain, Section 36. Indurated shale.
141. SW. end of Columbia mountain, inclusion in alaskite and rhyolite, principally former, Section 35. Black slate or indurated shale.
142, 143. Locality unknown. Silicified rock, indurated porphyry.

The Mojave Mining District of California.

BY CHARLES E. W. BATESON, NEW YORK CITY.

(Bethlehem Meeting, February, 1906.)

I. Location.

The Mojave mining district is situated in a group of small hills centering around Soledad peak, in the Mojave desert, Kern county, Cal. These hills are about 4.5 miles SSW. of Mojave, a railroad town at the junction of the Santa Fé and Southern Pacific railroads.

The town of Mojave is about 50 miles north of Los Angeles, to which it is connected both by telegraph and by rail. The distance on the Southern Pacific railroad is 102 miles, covered in 3 hr. and 45 min. by express train. A sketch-map of this region is shown in Fig. 1.

The Mojave district is relatively isolated. The only other mining towns in this section of California are Randsburg and Johannesburg, in a district (containing the well-known Yellow Aster mine) at the northern end of a low range of hills, which begins about 20 miles NE. of Mojave, runs nearly N. for 35 miles, and varies in width between 3 and 6 miles. As will be seen later, the origin and formation of these hills are not the same as those of Soledad peak.

II. Climate and Vegetation.

The climate and vegetation of the Mojave district are typical of the arid regions of the southwestern United States. During the winter and spring there is some rainfall, and the temperature is generally from 60° to 70° F. in the day-time. The rainfall in the spring is often very considerable for short spaces of time, and cloud-bursts are not infrequent. In summer there is continual sunshine, with temperatures, as a rule, exceeding 100° F. in the day-time, while 115° F. in the shade of a house is not at all unusual; but at night the temperature falls considerably. No rain falls during the summer. The vegetation

consists of a very little desert grass, a little grease-wood, sage bush and a few yucca trees.

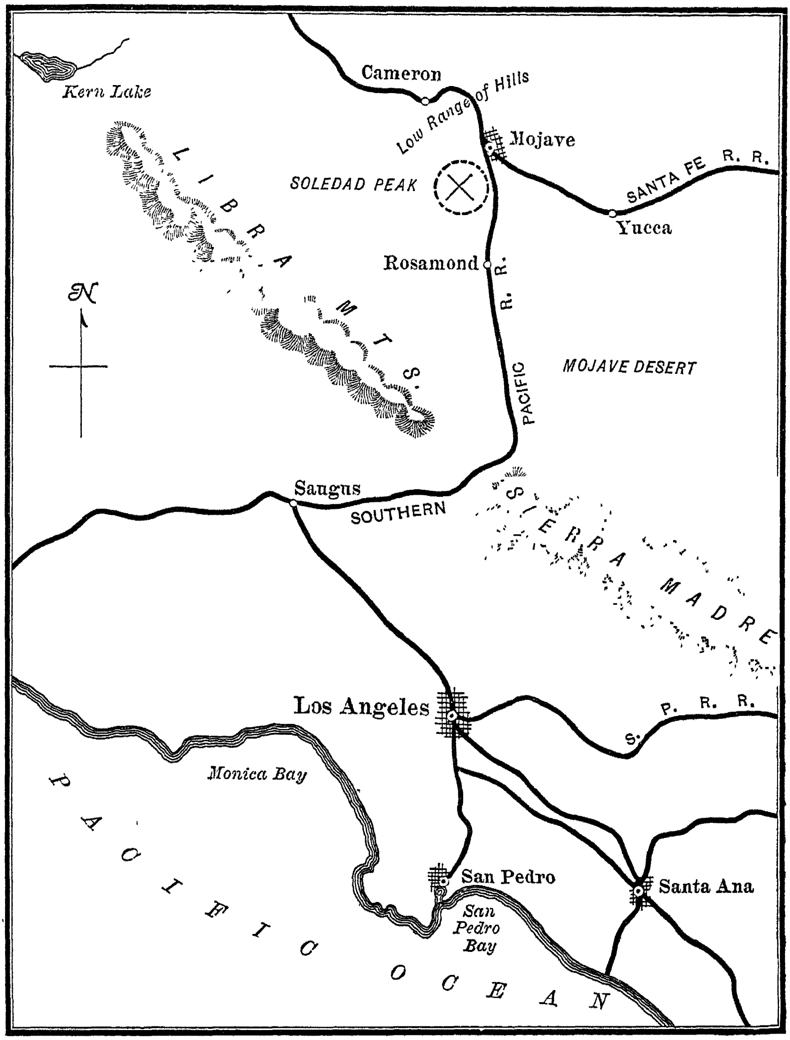

Fig. 1.—Sketch-Map of Southern Part of California, the Circle Showing Position of Soledad Peak.

III. Topography.

The topography presents low ranges of hills and isolated peaks of igneous or volcanic origin, which seldom rise more than from 1,500 to 2,000 ft. above the level of the desert. The

valleys, which are of much greater area than the hills, are the typical waste-filled interior basins characteristic of the desert.

Earlier in geological history, the valleys were deeper and the hills higher than they are now. The hills and ridges, being of igneous origin, were particularly susceptible to rapid disintegration, because the comparatively great change in temperature between day and night caused appreciable expansion and contraction in the outer crust of the rock, and the unequal rate of expansion and contraction of its constituent minerals produced stresses under which the rock-surface cracked and peeled off. The detritus, before any great amount of it could accumulate, was carried by the heavy rains into the gorges, where the water collected in great volume and swept it out into the valleys. This action exposed a fresh surface of the rock, to undergo the same cycle of disintegration.

The detritus was deposited at the mouth of the gorges as large alluvial fans, which increased in size until they met and merged into one another, forming detrital slopes. These slopes, extending outward and meeting the detrital slopes of adjacent ridges, formed detrital plains, which continued to grow until they completely filled the interior basin, and even overflowed it.

But little material, however, is carried out of these basins unless it be in solution. The water from the gorges soon loses itself in the coarse waste of the fans and plains, and does not reappear again, except, perhaps, when there has been a very heavy or extensive downpour, in which case it issues as innumerable springs at the head of a "crossing," *i.e.*, a gorge that connects one waste-filled valley with another.

The mouth of a crossing is shown in Fig. 2. The little hills, composed of bright red and white sandstone, are local and are sedimentary, while the surrounding hills are igneous. The age of these sediments I could not determine. The crossing is about 3 miles long. Water can be obtained in these crossings, even in the driest seasons, by digging a short distance, sometimes only a few feet; and also, in many cases, by digging at some distance from the mouth of a gorge that comes from the hills, especially if it be a main gorge, having branches and a pretty large steep cone at its mouth.

The porosity of the waste-filled valleys does not allow the water to go far on the surface, so that its burden, as it comes

from the hills, is entirely deposited on the alluvial fans. Subsequent rains move this deposit farther and farther out, until the line where two slope-plains meet is encountered, at which place the finer material is worked down towards the mouth of a crossing, so that the waste-filled valleys slope upward in every direction from the head of a crossing. The mouth of a crossing may or may not be one of the highest points in the waste valley into which it disgorges; but the general level of the valley from which it comes is higher than that of the valley into which it runs.

The local topography of the Mojave district shows a mass of eruptive rocks, whose highest and most northerly point, Sole-

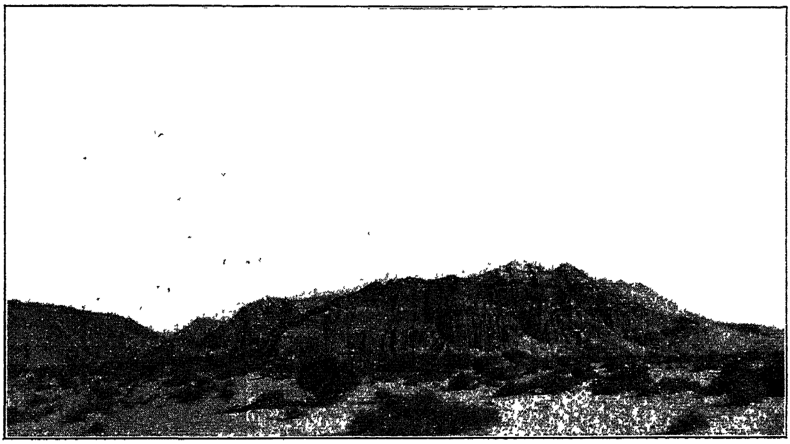

FIG. 2.—MOUTH OF A "CROSSING" IN THE MOJAVE DESERT.

dad peak, is about 4,000 ft. above sea-level and about 1,250 ft. above the general level of the surrounding desert. Mojave and Rosamond stations, on the desert, are 2,751 and 2,315 ft. above sea-level respectively.

A considerable number of much smaller hills are grouped south of Soledad peak, of which the miners consider them to be "spurs."

The map, Fig. 3, shows the generally circular to oval contours of all these hills.

Bowers hill, the only small one with a name, rises about 400 ft. above the desert, about a quarter of a mile northeasterly from Soledad peak, and is less than a mile long.

IV. Rocks.

Five varieties of igneous rock appear in the district.

1. *Granite.*—This is the oldest formation exposed, and prob-

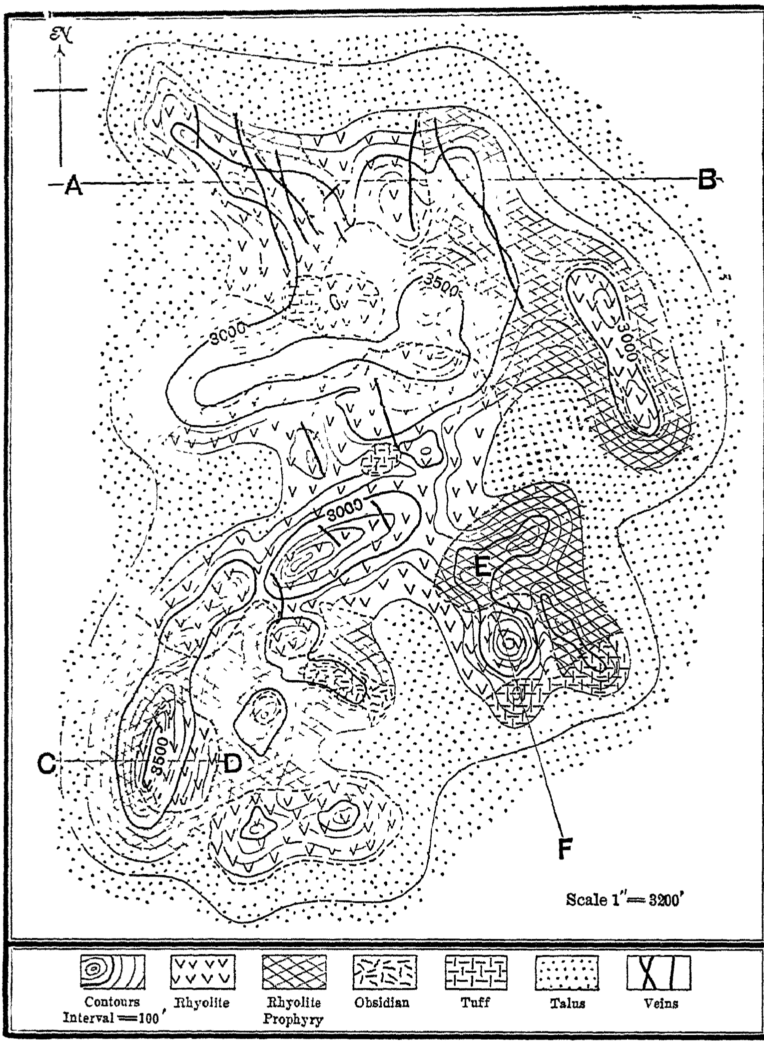

Fig. 3.—Topographical and Geological Map of the Mojave District, Kern County, Cal.

ably underlies the entire district. It shows at the surface along the northeast flank of Bowers hill. It extends to the southwest and does not re-appear in the mineralized zone. It is seen,

however, about 5 miles SSW. of Soledad peak, where it underlies some low hills of tuff.

Hand-specimens of this granite show a coarse-grained quartz-orthoclase-albite rock, containing a very small quantity of biotite.

2. *Rhyolite-Porphyry.*—This rock, lying upon and in contact with the granite, has by far the greatest exposure of all the rocks of the district, as shown in Fig. 3, and is the most important by reason of its quantity, and because it is the country-rock of the veins.

Both in hand-specimens and under the microscope, it presents a much decomposed appearance. Quartz, often showing a decidedly hexagonal outline, is the only mineral distinguish-

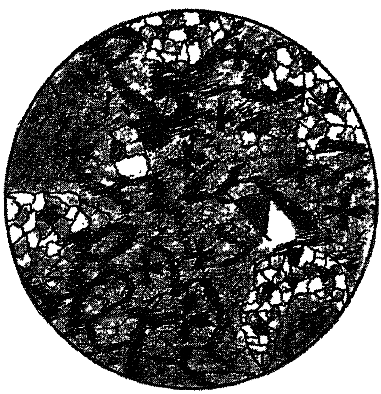

Fig. 4.—Rhyolite-Porphyry. Greatly Altered.

able to the naked eye. In hand-specimens the rhyolite-porphyry is nearly or quite white. In thin sections under the microscope the rock is seen to be very extensively altered, as shown in Fig. 4. Because of the complete obliteration of all traces of structure, quartz is the only original mineral recognizable. Quartz occurs, also, in considerable amount as a secondary mineral. Kaolin, showing slightly developed spheroidal structure, is present in large quantity. Secondary calcite is probably present, but in no place is it developed sufficiently to be identified. Neither in hand-specimens nor in thin sections does the rhyolite-porphyry show lines of flow.

3. *Rhyolite.*—This rock, in sheets and patches, overlies the rhyolite-porphyry and forms the summits of many of the hills,

as shown in Fig. 3. A few veins outcrop in the rhyolite, but they have not been developed, even slightly.

In hand-specimens, the rhyolite is dirty brown in color, and greatly resembles devitrified obsidian, having strongly marked flow-lines. In thin sections under the microscope, as shown in

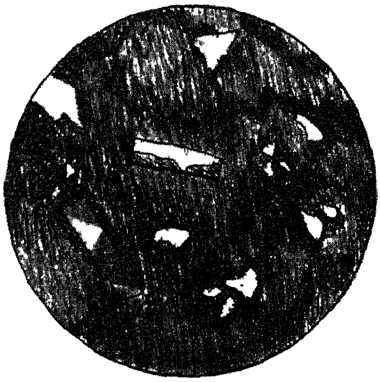

FIG. 5.—RHYOLITE GREATLY ALTERED, SHOWING ORTHOCLASE TWIN CRYSTAL REPLACED BY QUARTZ.

Fig. 5, it is seen to be very much decomposed and similar to the rhyolite-porphyry. Quartz, the only recognizable mineral, is both primary and secondary. In many cases the secondary quartz has replaced feldspar crystals, molecule by molecule;

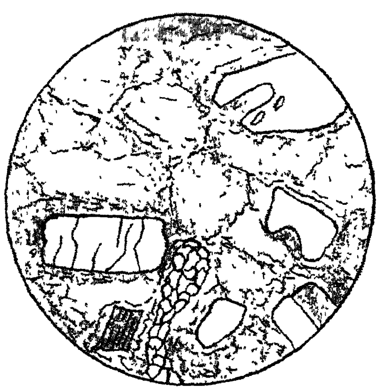

FIG. 6.—RHYOLITE, SHOWING LINES OF FLOW.

and in one case (the crystal near the center of Fig. 5) it has even preserved the orientation of an orthoclase twin crystal. No other mineral is recognizable.

Fig. 6 shows a thin section of a nearly fresh specimen of this rhyolite. In hand-specimens it is almost white, of fine texture,

and shows faint flow-lines. In the thin sections it is seen to be a devitrified rhyolite, showing phenocrysts of orthoclase, biotite and quartz. The biotite and orthoclase have no unusual features, and the quartz occurs both primary and secondary. The primary quartz originally had well-marked crystal outlines that later became indistinct in most of the crystals through re-absorption of the silica; while, in others, it remained fairly well preserved. The secondary quartz is in minute, interlocking pieces of no crystal outline and of varying orientation.

4. *Obsidian.*—In hand-specimens, the obsidian is black in color and full of small holes, about an eighth of an inch in diameter. Thin sections are nearly opaque, and show nothing except that it is greatly devitrified. It is probably a more quickly-chilled part of the rhyolite, and not a distinct flow. It occurs only in one place, as a capping to one of the smaller hills, and is underlain by the rhyolite-porphyry described above, and shown in Fig. 3.

5. *Tuff.*—The tuff, composed of volcanic ash, is now so much decomposed that very little can be made of its mineralogical character except that quartz and, at rare intervals, a fragment of feldspar are recognizable. In hand-specimens it generally varies from a dirty white to a pale, greenish color, and appears rather fine-textured for a tuff. It occurs in three small patches, overlying the rhyolite in two cases, while in the the third it seems to overlie a contact between the rhyolite and the rhyolite-porphyry. Its actual contact was not seen on the rhyolite-porphyry, being covered by the talus, although it was seen on the rhyolite. The map, Fig. 3, shows it resting in part on the rhyolite-porphyry, because outcrops of the rhyolite-porphyry came very near it, and there was no intermediate talus of rhyolite. The detritus and waste of the hills are shown in Fig. 3 as surrounding them.

V. Origin of the Rocks.

There are two tenable hypotheses in regard to the relation between the rhyolite-porphyry and the rhyolite.

1. According to the first, the rhyolite is the outside quickly-chilled portion, and the rhyolite-porphyry is the more slowly-cooled portion, of a volcanic outburst.

2. According to the second, there have been several erup-

tions, at least two of which are observable. Of these, the first produced the rhyolite-porphyry (which may or may not have been finer-grained on its surface), and was followed by a period of quiescence, during which considerable erosion took place, and after which the second eruption produced the rhyolite, and, in its expiring stages, the volcanic tuffs.

The evidences to support the first view (that of a single flow) are:—

In several places the rhyolite grades from a rock which resembles the obsidian, shown in Fig. 5, having the same dirty brown color, into a nearly porphyritic rock almost white in color, shown in Fig. 3. Then, after an interval of from 150 to 200 ft. covered by talus, the typical rhyolite-porphyry appears. At no place was I able to find exposed a continuous gradation of one into the other. The mineralogical characters of the rock serve in a way to support this view. Both the typical rhyolite, shown in Fig. 5, and the typical rhyolite-porphyry, shown in Fig. 4, are too greatly decomposed to give clear evidence; and the only satisfactory thin sections, shown in Fig. 6, came from a rock that is almost porphyritic in texture, and is clearly a part of the rhyolite.

In favor of the second hypothesis, namely, that there were two separate and distinct flows, the following facts may be regarded as evidence:—

In one spot, on the Echo claim, a direct contact was found where the rhyolite-porphyry is typically developed, as is also the rhyolite. The rhyolite-porphyry occurs as a small circular outcrop, about 2 ft. in diameter, which is entirely surrounded by a breccia, from 1.5 to 3 in. thick, colored brown and containing a great deal of quartz, followed by the rhyolite. I think that the presence of the breccia precludes the view that this outcrop is a boulder that had become imbedded in the rhyolite. But whether it is a boulder or rock in place is unimportant, since in either case its occurrence shows that the rhyolite-porphyry is older than the rhyolite.

One of the smaller hills was capped with rhyolite in the form of black obsidian, below which all outcrops were concealed by talus until near the base, where rhyolite-porphyry was exposed. These two rocks, the rhyolite and the rhyolite-porphyry, were sharply contrasted and could easily be distinguished. The

talus, however, contained only the typical examples of each, there being no transition-forms. From this occurrence the inference was reached that they represented distinct and separate outbreaks.

The dome-like shape of the hills, with the rhyolite as a cap, shown in Fig. 3, seem to indicate that a cone of this harder, more-resistant rock came up through the rhyolite-porphyry in necks and overflowed on the surface, and that these necks have resisted weathering better than the more easily disintegrable rhyolite-porphyry. The mineralogical characteristics of the rocks, however, do not strongly support this hypothesis. Practically all that can be said is that the primary quartz in the rhyolite generally shows considerable re-absorption, while that of the rhyolite-porphyry does not; moreover, the rhyolite shows flow-lines, which are not observed in the rhyolite-porphyry. The mineralogical characters of the rocks would, on either hypothesis, be very similar, since the magmas were in all probability identical, being doubtless derived from the underlying granite. Hence, the relationships in the field are the strongest and most trustworthy evidence, and they indicate, with a high degree of probability, two flows. These relationships are illustrated by Fig. 7.

That the Mojave district is an isolated one, and that the origin of all the desert hills is not the same, or even due to similar causes, is shown by contrasts presented in the Randsburg-Johannesburg area. That district, as already observed, occupies the northern end of a long, low ridge rising from 1,000 to 1,500 ft. above the general level of the surrounding desert. The back-bone of the ridge is diorite, which is flanked on both sides by heavy beds of metamorphosed hornblende schist, dipping in places as much as 45° away from the center or axis of folding.[1]

VI. Veins.

Almost all the veins now known are on the northern flank of Soledad peak, and their strike is generally a little south of west, as is illustrated in Fig. 3. The principal and largest veins are operated by four companies. Beginning at the west, the Echo Mining Co. works the Echo, Grey Eagle and Star-

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[1] 13*th Report of the State Geologist of California:* Gold—Kern Co. (1895–96).

light veins; about 0.5 mile farther east the Queen Esther Mining Co. works the Queen Esther vein; and next to the Queen Esther is the Karma Mining Co., working the Karma vein.

The Exposed Treasure vein on Bowers hill is worked by the Treasure Mining Co., a New York concern.

The Echo vein has the normal strike and dips to the northeast, as shown by Fig. 7. Its outcrop can be traced for 1,800 ft., and

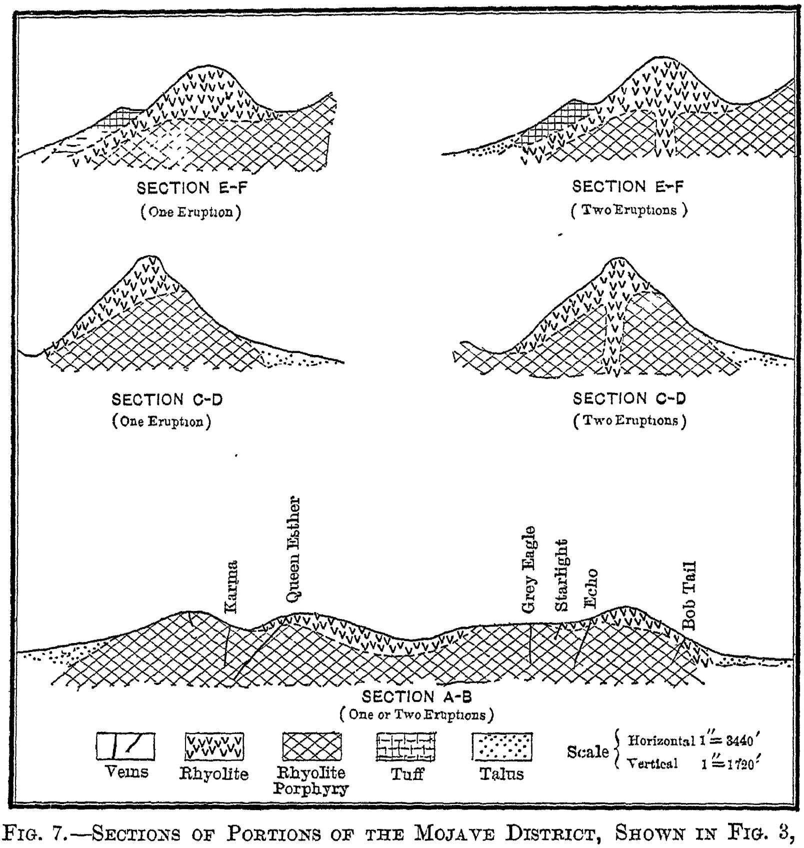

FIG. 7.—SECTIONS OF PORTIONS OF THE MOJAVE DISTRICT, SHOWN IN FIG. 3, ILLUSTRATING THE PROBABLE RELATIONSHIPS.

throughout its entire length a rather constant width of about 3 ft. is preserved. This general width also persists in depth as far as the vein is known—something over 200 feet. The gangue, or vein-filling, is quartz of the variety generally known as "sugary." In general, the line of demarcation between the quartz-filling and the country-rock is sharp, although in places the walls seem to be much shattered for a few feet on each side of the vein, the

quartz having filled any spaces and cracks that were formed. No gouge-seams, slickensides or other evidences of extensive movement after the quartz-filling had been introduced were observed. In places next to the country-rock, and even in the quartz itself, patches of kaolin are found. The wall-rock is silicified rhyolite-porphyry, which is very hard, and difficult to mine. The values are in gold and silver. The gold occurs free in very minute scales associated with pyrite. Along the outcrop and near the surface the gold is almost all free. The silver occurs as horn silver (cerargyrite) in small specks along and near the outcrop, changing to argentite in depth. The entire vein is mineralized to some extent. The workable portions, however, occur in irregular bunches, which may have considerable dimensions, both in length and depth; that is, from 100 to 200 feet.

The Starlight vein, distant about 300 ft. east of the Echo, is parallel to it in strike, and dips to the northeast. Its outcrop cannot be traced for the entire distance. Its average width is about 3 feet. The vein-filling, the relationship between the vein and the wall-rock and the mineralization, are the same as in the Echo. The major part of the value is in silver.

The Grey Eagle vein commences near the northern end of the Starlight and diverges from it in a southerly direction, until about 100 ft. or more from it, and then turns parallel to it. In general the dip is 90°, from which, however, it varies, sometimes to one side and sometimes to the other. The vein connects with the Starlight at increasing depths towards the south. Its average width is a little less than 3 feet. The vein-filling, and the relationship of vein to the wall-rock and the mineralization, are the same as the Echo and Starlight, except that the gold-value is greater than that of the silver. At the one place developed, where the two veins join, shown in Fig. 8, considerable kaolin occurs, and the values are not so high as on either side of the connection. The quartz-filling, however, shows no discontinuity in passing from above the connection to below it.

The mine-management believes that the Grey Eagle crosses the Starlight vein, basing the belief on a small length of outcrop found to the west of the Starlight vein, having a strike nearly the same as that of the Starlight vein, and being in a general line with that part of the Grey Eagle vein which joins

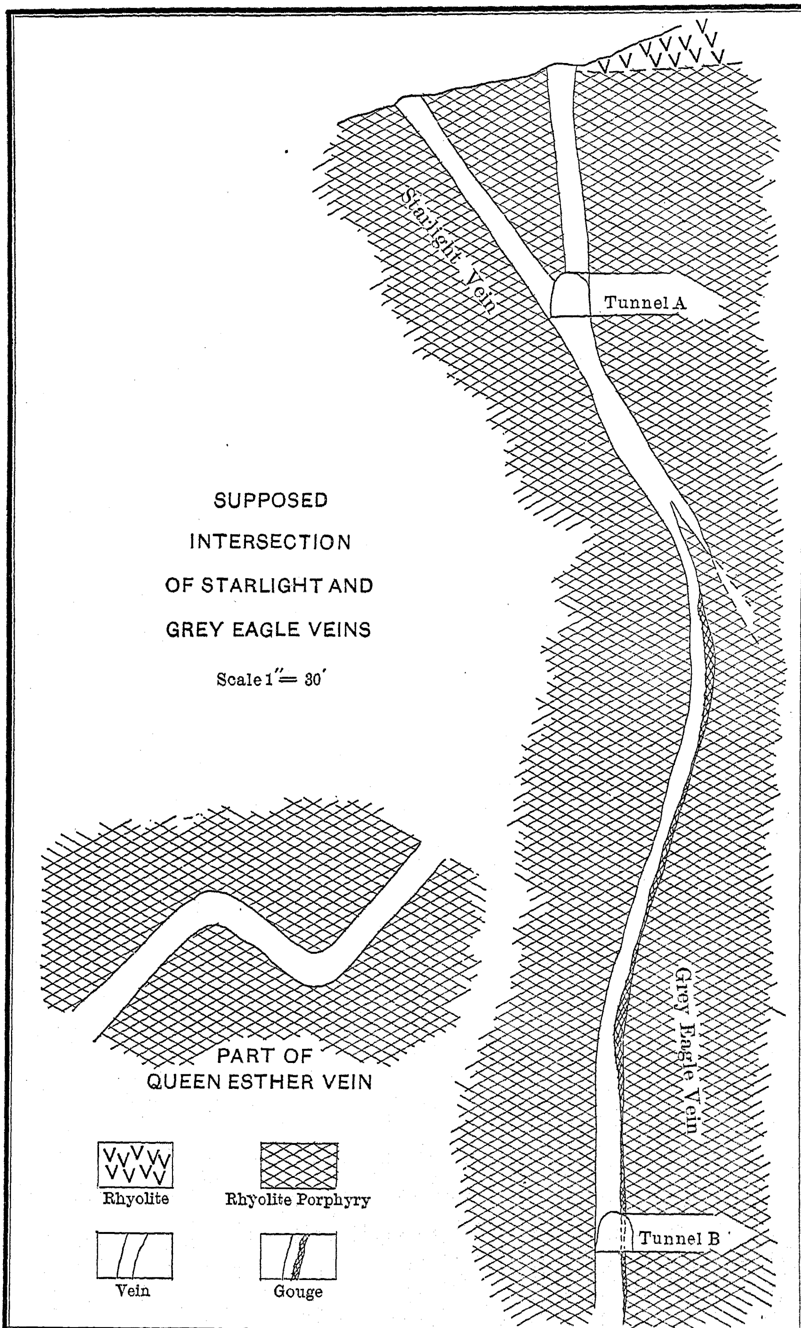

FIG. 8.—SECTIONS OF STARLIGHT, GREY EAGLE AND QUEEN ESTHER VEINS.

on the outcrop. It is believed that the Starlight has faulted the Grey Eagle.

The Queen Esther vein runs more nearly north and south than any of the others. It dips about 60° to the east. The vein-filling is sugary quartz. The country-rock is greatly decomposed rhyolite-porphyry. The mineralization is similar to the Echo. The width of this vein is generally from 5 to 6 ft., but it varies a great deal, in some places being nearly 12 ft., while in others it is only about 3 feet. A remarkable feature of this vein is the rolls that it takes as it gains in depth. This is shown in Fig. 8. At the present stage of development three of these rolls or folds are known. The first one encountered in depth is the smallest, and the third the largest. They dip along the vein about 10° to the north. The largest one displaces the vein 30 ft. horizontally and 10 ft. vertically.

The Karma vein, dipping about 85° to the east, has the prevailing strike of a little south of west. Its outcrop can be traced continuously for about 1,000 feet. It varies in width from 2 to 5 ft., although, in places, the mineralized zone reaches considerable width. In the case of the "glory hole," it is about 40 ft. to the vein-stuff, which fills cracks and fissures extending into the country-rock on both sides of the vein proper. At one place on the outcrop, about 200 ft. in length, this vein, together with the numerous small branches it there possesses, reaches a width of between 60 and 80 ft., but it is reported not to be well-mineralized at this point.

The gangue is sugary quartz, and the mineralization is the same as that described above. Bromyrite was reported to have been abundant in one of the stopes, now worked out. A small piece of quartz, still adhering to one of the walls, was obtained, showing small specks of the reported bromyrite. On careful investigation (the specimen was so small that entirely satisfactory results were impossible) the green specks were found to be in all probability copper stains, and not bromyrite at all. The character of this mineral as determined are:—color, dark green; luster, dull; hardness, from 4 to 5; opaque, even under the microscope. Very rich ore was obtained from this stope. The country-rock of this vein is rhyolite-porphyry.

The Exposed Treasure vein is on Bowers hill. I did not examine this vein, but from hearsay I present the following:—

The vein maintains the usual strike of all the veins of the district, but departs from the rule by dipping to the west. The wall-rock for a distance is rhyolite-porphyry, and then the vein follows the contact between the granite and the rhyolite-porphyry. The mineralization follows the general rule, and the pay ore is very rich but extremely irregular.

A small vein on the southwestern flank of Soledad peak, thought in all probability to be an extension of the Starlight (see Fig. 3), is interesting, because its wall-rock (see Fig. 9) shows a double brecciation. It also shows pyrite in the cementing material of the second brecciation. The first cementing material was mainly kaolin, with some secondary quartz. The

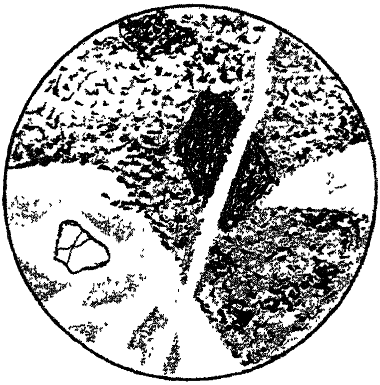

FIG. 9.—SECTION OF WALL-ROCK OF VEIN ON SOLEDAD PEAK, THOUGHT TO BE A CONTINUATION OF THE STARLIGHT VEIN.

second cement comprised secondary quartz and considerable kaolin, in places with pyrite, which carries considerable gold.

Whatever the history of the volcanic action may have been, it is quite evident that the veins were not formed until all volcanic action was over.

Almost all the known veins are on the northern flank of the group of hills, and these are easily divisible into two main groups:—one, the Echo group, to the west; and the other, the Queen Esther-Karma group, to the east. The Exposed Treasure and others, perhaps not yet found, make a third but less important group on Bowers hill. The general course of all these veins is a little west of north.

Both of the first two groups seem to have a focus where all

of the veins in each group will meet, provided they preserve their present dip. In the Echo group, this focus is about 700 ft. below the general level of the desert, and in the Queen Esther-Karma group it is from 400 to 500 feet..

All these veins are typical quartz-veins, containing quartz and minerals derived from the decomposition or alteration of the country-rock, and deposited in fissures, caused by contraction on cooling; in this case the cooling has been from fusion. In some of the veins the brecciation of the wall-rock is clearly observable, and in one case a double breccia—a brecciated breccia—is shown.

In the Queen Esther vein rather remarkable rolls are found, which were not caused by any cross-fissures or faults, so far as could be observed, but were probably due to the original fissuring of the rock-mass by cooling, which has since remained undisturbed; that is, no vertical movement of the walls has taken place.

From the above description it appears that the fissures were originally formed by cooling, followed, in some cases, by slight movements of the rocks which caused the production of breccia by the attrition of the walls over each other. This result has happened, in one observed case, at least twice. In other cases there was no movement whatever, and in no case observed did the movement amount to more than a few inches. Accurate observation concerning the amount of movement in volcanic rocks, especially if they have been badly altered by decomposition, is extremely difficult, if not impossible.

A movement of only a few inches, or at most a few feet, is inferred here because there has clearly been no movement in some of the veins; and while there is brecciation of the walls in other veins, there is not enough variation in the width of any one of them to indicate extensive movement.

The quartz-filling of the veins and the silicification of the walls certainly did not take place until all movement had ceased, and since then there has been little or no movement, so far as has yet been observed.

In one or two veins there is a little gouge occuring irregularly on one or the other of the walls, which indicate some slight movement of that wall. This gouge is nearly pure kaolin, and in no observed locality did it cross the vein of

quartz to the other wall; moreover, it was not great in amount; hence, at best, the movement was only a slight one. The Starlight-Grey Eagle junction may or may not be a cross-fissure. If it is, the fissuring was previous to the filling of the vein. The values are gold and silver. The gold occurs native in minute specks, and to a small extent in pyrite. The silver occurs mainly as a black sulphide, argentite, with cerargyrite in small grains sparingly disseminated in places near the outcrops. The values are not at all regularly distributed, but occur in irregular patches of no definite outline, and which vary in size up to a few hundred feet along the vein to a hundred or more in depth. The width of the veins varies from a few feet up to 12 feet. Mineralized zones occur, however, of much greater width.

VII. Summary.

The Mojave district is of volcanic origin, and has been formed by two eruptions, the first of quartz-porphyry and the second of rhyolite. The underlying country-rock is granite. After all volcanic action had ceased, fissuring took place, followed by deposition, from a rising solution, of the quartz with the gold- and silver-minerals, thus forming the veins.

The eruptives are very greatly altered, so much so that, with few exceptions, their original minerals are changed beyond recognition. The general form of alteration—the original minerals being quartz, orthoclase, some other feldspar, probably albite and biotite, with pyrite and perhaps some other minerals as accessories—is that which is typical of the belt of weathering in arid regions; namely:—

1. The feldspar was altered to kaolin and quartz, with the loss of potassium, calcium or sodium, as the case may be.
2. The original quartz remained as such.
3. The biotite was altered to kaolin with the probable formation of serpentine and gibbsite, although no gibbsite was observed.
4. The pyrite, gold- and silver-minerals went through the orthodox cycle of changes. The values in the early history of the veins probably were very evenly distributed.

Since the formation of the veins, however, considerable erosion has taken place, with the consequent lowering of the outcrop and also of the water-level. Secondary enrichment has

taken place in portions of the vein by the well-known phenomena of downward concentration, caused by the leaching of the values from the upper portion of the vein, and their deposition at a lower level, at or near the ground-water level.

No mine, with the possible exception of the one on Bowers hill, has reached the ground-water level; hence, nothing can be said of the veins below that level, or of the further distance necessary to sink in order to reach it.

There is no reason why the mineralization of the veins should not continue into the granite, since in this district the wall-rocks, being of a like, if not identical, chemical composition, and varying only in the degree of their crystallization, possession or non-possession of flow-lines, etc., can have had no differential effects on the deposition of the ores in different parts of the vein. The portions enriched by downward concentration, which, at present, form the pay-portions of the veins, can certainly be expected to end when the ground-water level is reached. Whether it will pay to work the veins below that level—at present unknown—is a problem which can be solved only by actual trial.

Notes on Southern Nevada and Inyo County, California.

BY H. H. TAFT, DENVER, COLO.

I. Introduction.

The mining possibilities of the volcanic area south of Belmont, Nye county, Nevada, have long been known. Some of the old-time prospectors knew that gold existed there. Its remoteness from any source of supplies, its long distances from water, the absence of game, and more, perhaps, the lack of grass for animals to subsist upon, has made this an unattractive region in which to search for mines.

The decline of the Comstock mines, the exhaustion of sundry large and rich ore-bodies, the high cost of mining, marketing, and, particularly, the ruinously high freight-charges upon refractory ore that had to be shipped to distant smelters, have kept investors out of Nevada for the last few years; and mining people have hardly yet awakened to the importance of Tonopah, Goldfield, and perhaps the newer and less developed districts.

The discovery of Tonopah by J. L. Butler, who located the Mizpah claim in May, 1900, and the fortunes soon realized there, attracted many people. As the boom declined, many went away, some scattering out into the surrounding country, and the population is now about 7,000. In the fall of 1902 a discovery of gold was made 23 miles south, in what is now known as the Sandstorm group, 4 miles NW. of Goldfield. In the winter of 1903–4 the Combination, January and Florence mines were discovered, and shipments of high-grade ore soon followed. In January, 1905, there were 10,000 people in Goldfield.

In June, 1904, rich gold-ore was found 85 miles SE., at the foot of the south end of the Kawich mountains, but this discovery was kept quiet until a re-location could be made.

On August 10, 1904, the Bullfrog claims, and a month later the Ladd mountain and neighboring claims, were located. The Shoshone group was located September 24. This district is

from 60 to 80 miles SE. of Goldfield. In September, 1904, there was a stampede for Bullfrog and Gold Crater—the latter a small area 21 miles east of Goldfield. Two mining districts, called Beatty and Bullfrog, were organized under the laws of Nevada. Later, overflow migrations poured into the old and abandoned districts of Lida (or Allida), Tule Cañon, State Line and Silver Peak, and others more remote.

During the winter of 1904–5 the desert seemed full of people. All sorts of outfits traversed unfrequented roads—men afoot and alone, "burro men," carriages, wagons and automobiles. The inevitable reaction of such a furor is no doubt deplorable; yet the rapid development of any new mining region depends upon the excited "tenderfoot" rather than the conservative mine-operator. At Goldfield, it was "a sight to see." There were hundreds of people walking over the hills, many with a canteen of water slung over one shoulder, while a small iron mortar hung to the other, and a pestle, a pick and a 5-in. frying-pan constituted the equipment for sampling, grinding and testing. The rock is soft and the gold at the surface is free.

There is no very good map of this region. The best is that of the U. S. Geological Survey[1]; but this and the Land-Office maps are incorrect, particularly in the topography between "Lost valley," the northwest arm of Death valley, and Owens Lake. A correct map of Inyo county, Cal., has been made by the County Surveyor. A very useful map, particularly of the country farther south, is issued by Mr. Crowell, of Vegas, Nev.

The high Panamint range is usually mapped with about twice its actual length. It ends at latitude 36°30′ north. A wagon-road, from Furnace creek in Death valley to Ballarat in the Panamint valley, follows around this mountain at its base. The geological maps would be far more useful to prospectors if the older tertiary volcanics were separated from the recent ones. A good map showing the potable waters would save much suffering, and perhaps some lives, this summer. The springs should be marked by the Government. So far this year about 30 lives have been lost on account of thirst in that desert region.

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[1] Bulletin No. 208, *U. S. Geological Survey* (1903).

Tonopah is 6,000, Goldfield 5,500, and the Bullfrog region from 3,500 to 4,000 ft., above sea-level. The climate at Goldfield is much the same as that at Pueblo, Colo., except that the rain-fall is less than half. The topographical variation is not great in Nevada, the summits of the mountain ranges are rarely more than 2,000 or 3,000 ft. above the surrounding deserts. Inyo county, Cal., is different, being remarkable for deep valleys and high precipitous mountains. The altitude of Owens Lake is 3,575 ft., and the Sierra Nevada, a few miles west, reaches an elevation of 14,522[2] ft., above sea-level. The Panamint peaks rise to 11,000 ft., while Death valley, opposite these peaks, and but a few miles east, is below sea-level. On the west, the Panamint valley is 1,100 ft. above tide, while Saline and Butte valleys are not far from sea-level. Roughly speaking, 600 ft. in elevation is equivalent to 1 degree in latitude; moreover, a deep valley has not the circulation of air that prevails on the table-lands. Hence this is a region of extremes of wind and calm, heat and cold, both diurnal and annual. The storms of winter seem to blow through one and to take all warmth away; yet on a summer day, without any shade, down in one of these deep valleys protected by high mountains from the prevailing winds, it is hotter than in any other part of the American continent. The maximum temperature at Furnace Creek ranch, in Death valley, is said by those who live there to be 127° F. The extraordinary amount of detritus brought down through every little gulch indicates terrific cloud-bursts.

Life would not be so intolerable in these valleys in the summer season if our people would learn more of the Mexicans. Americans even go across the border into Mexico and farther south with their light board houses, low ceilings and thin roofs. Thick stone or adobe buildings with high ceilings and thick or double roofs are always comfortable. The good climate of the whole year in Tonopah and Goldfield is much appreciated by the mining men who have come from the tropics, Alaska, British Columbia, and the higher altitudes of Colorado.

Montezuma mountain, 8 miles west of Goldfield (altitude 8,000 ft.), is ever green with piñon and scrubby pine, with a little cedar and juniper around the edges. It is remarkable how closely one can estimate the elevation through the entire

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[2] Mt. Whitney.

Rocky Mountain region by noting the vegetation. The Montezuma Mountain timber-belt extends SW. to the White mountains, and to the Fish Lake range. The north end of the Grapevine range is also covered with timber. Cord-wood sells in Tonopah and Goldfield at $16 per cord; at Bullfrog the price is $25. Below the timber there is considerable sage brush, with a few cacti and some yucca palms, locally known as Joshua trees. The desert is often green with several varieties of desert brush having different local names,—a short, stunted growth of no value, which gives the valleys the appearance of being more fertile than they really are. A strange feature is the almost entire absence of grass. At the head of the Amargosa and southward, "creosote" brush and other desert growths are the same as one sees in northern Chihuahua, western Texas, and in New Mexico, which is good grazing country. Old-timers say that this was not so once, but that several years ago a drought killed all vegetation that could be used as fodder. Along the water-courses, such as the Oasis valley, at Ash meadows, in the Death valley, Panamint, and others where there is water, salt and wire grass present a meadow-like appearance, but will barely keep cattle from starving. Along the water-courses, willows, cottonwood, and to the south, screw-bean and mesquite, grow. The last, which is an excellent fuel, is by common consent left to the Indians. Sometimes there are in the valleys large areas devoid of vegetation, with the ground so hard that a wagon leaves but a slight track. Along the low ridges the wind has blown away the soil and arranged the pebbles so as to appear like a mosaic.

The traveler usually takes the "Overland Limited" to Reno, Nev., then the Virginia and Truckee railroad 41 miles to Mound House, the Carson and Colorado 137 miles to Sodaville, and the Tonopah railroad 66 miles to Tonopah. The Carson and Colorado was a narrow-gauge road with light rails and limited equipment, completed to Keeler, Inyo county, Cal., in 1881. It was a barren investment until lately, when the Southern Pacific Co. obtained control of it, and of the narrow-gauge road from Sodaville to Tonopah, reconstructing both to broad gauge, just in time to reap the benefits of the Tonopah rush. During the past winter this whole road has been swamped with freight. For 6 months there were from 500 to 1,000 cars

awaiting trans-shipment in the various yards near Reno. From Tonopah to Goldfield both stages and automobiles were formerly running,—the latter making the distance of 27 miles in 2 hours. All this is now changed, and the traveler can leave the main line of the Central Pacific in a broad-gauge car that will take him through from San Francisco to Tonopah and Goldfield.

Surveys have been made and there is much talk of railroads from the south. A factor in this situation is furnished by the large deposits of colemanite (calcium borate) between Amargosa and Death valleys. A railroad-route presenting no special difficulty runs from Vegas on the "Clark" road to Goldfield, via Beatty. The distance to Ash Meadows is 90 miles; thence it is 45 miles up the valley to the Bullfrog district, and 35 miles diagonally across (south) to the most developed of the borax-mines.

The outfit for a trip through this section requires, as a usual rule, 1 lb. each of vegetable and animal food per day per man, and 14 lb. of hay and 12 lb. of grain per horse per day. A larger amount of alfalfa, with a smaller amount of barley, can be fed. Mules are preferable to horses, because they are more hardy and eat and drink less. In a country where evaporation is so great (an acre of tanks will evaporate 1,700 gal. per day), a team of horses will require about 15 gal., say 120 lb., of water per day, and in the heat of the summer, more. Where there is running water in the winter, there is nothing but a dry "arroyo" in the summer; indeed, in a channel where there is a stream of running water in the morning sufficient for stock, it may be dry and even dusty at sundown.

A good assaying-equipment sufficient for 1,000 assays will weigh 500 lb. and require 5 cases of gasoline. A portable balance sensitive to 0.005 mg. can be had, and is best in that it enables one to reach a desired degree of accuracy with less of fluxes and a smaller weight of crucibles.

The people are very kind about giving information as to water and roads; but such information is often inaccurate. Nye county, Nev., has had sign-boards put up at cross-roads; and some of the freighters, also, are thoughtful enough to leave some mark or sign.

II. From Vegas to Ash Meadows.

For about 90 miles the road follows a succession of desert valleys. The Las Vegas mountain, north, and the Charleston mountain, south, both of Carboniferous limestone, are little disturbed, but the small spurs west are considerably contorted. About 12 miles NE. of Ash Meadows, north of the road, the limestone beds are tilted and underlain by quartzite; and a little south of the road there is a small area of basaltic lava which has overflowed a recent volcanic tuff. For a distance of from 2 to 20 miles SE. of Ash Meadows there is considerable quartzite. Although prospectors report some lead and copper, the region is unattractive on account of the absence of eruptive rocks. On the NW. slope of Charleston mountain are two old mining districts, the Montgomery (now known as the Johnny) and the Stirling. These districts, abandoned for some years, have now taken on new life. The ore is gold-quartz, with a little pyrite and chalcopyrite.

III. The Amargosa Desert.

This desert valley is about 100 miles long, and forms with Death valley a long narrow U, extending NW.–SE. The upper end, 4,000 ft. above sea-level, is formed by the joining of the Grapevine and the Amargosa mountains; the former is the northern boundary of Death valley, and the latter contains the Bullfrog mining region. The Amargosa is cut through by the Oasis, a narrow valley in which are numerous springs and a little running water. Opposite the Oasis, the Amargosa is 12 miles wide; farther south it widens rapidly. Between the mouth of Forty Mile cañon and the Funeral range it is 30 miles wide. Here the road is so dry and sandy that freighters have to "double," and then can only travel at half the usual rate of speed. The roadside-graves and skeletons of draught-animals are mute witnesses of the hardships here experienced. An enormous sand dune shows that the contour of the mountains has some peculiar effect upon the winds.

Forty-five miles SE. of the Oasis valley is a series of springs. The general locality is known as Ash Meadows. Here the valley is 15 miles wide from the meadows to the Funeral range. In a distance of 6 miles there are four springs flowing about 50

miners' inches of water each, and a number of smaller ones. The temperature of the water in the larger springs is 76° F. and in one of the smaller ones 94° F. All these waters carry a large proportion of sodium carbonate, a remarkable amount of aluminum, a little borax and a small amount of sulphates. From the southerly spring a stream flows for a distance of from 5 to 20 miles, depending upon the season. Below these springs there are large areas of apparently good meadow-land; but the rushes, salt and wire grass are of little value as fodder.

Pahrump ranch is 30 miles SE. of Ash Meadows; and 6 miles farther is the Manse. These ranches are veritable oases; and the extraordinary market which they enjoyed last winter for fodder, vegetables and fruit was a godsend to their owners.

IV. Bullfrog Mining District.

At the head of the Amargosa desert, the Grapevine range, 3,000 ft. above the valley east, and 7,000 ft. above Death valley west, is flanked east by recent volcanic tuffs. Along the summit occur limestones and quartzites dipping east, and a lime-conglomerate, carrying granite, diorite and quartz, such as occur many miles north.

The Amargosa range, lying between the south end of the Ralston desert and the NW. end of the Amargosa, is formed by a series of tuffs superimposed upon limestone. The various members of the volcanic series occur with the regularity of sedimentary strata, and the upper (consequently more recent) ones are highly colored. They dip from 10° to 20° NE. An extensive block-faulting has exposed the edges of the various flows, particularly from the west, the escarpment being on that side. About 2 miles west of the Bullfrog mine is a small hill of gneiss, overlain by strata of chloritic slates, quartzites, limestones, and tuffs, dipping flat to the east.

On August 10, 1904, two claims were located as Bullfrog No. 1 and No. 2. From August 10 to September 14, 1904, a large number of claims were located on what are now known as Ladd and Bonanza mountains, about 3 or 4 miles SE.

The Bullfrog Mining Co. was formed to take up a group of Ladd Mountain claims; and later, the first discovery of the district was transferred to a new corporation, The Original Bullfrog Mining Syndicate. A mile south of the Original Bullfrog

property some claims were staked off, surveyed and sold as lots under the name of "Bullfrog town." Amargosa, a mile further south, aspired to be the metropolis. Out on the desert west of Ladd mountain the same thing was done; and the "town" was named Bonanza. Four miles from the latter place, in the Oasis valley, on the bank of a stream of running water, the town of Beatty, named for a ranchman living a mile above, was laid out, and soon became the most populous place. Three miles below Beatty a group of tents bore the name of Gold Center. In March, 1905, in a cove made by the desert in the Amargosa mountain, between Bonanza and Ladd mountains, Rhyolite was laid out, and Bullfrog and Bonanza moved to it. This place is 5 miles from water.

About 10 miles SW. of where Oasis valley breaks through the Amargosa range, the cliffs are of limestone which pitches west and is soon buried beneath the soil and the volcanic tuffs that have probably borne it down. In several places the contact is exposed, and there are evidences of a flow of water not accompanied by a siliceous deposition, except in and near the Bullfrog claims. Between Ladd mountain and the Oasis creek there is a place where there has been a considerable spring on the contact. Boulders of granite 3 in. in diameter are scattered over an area 100 ft. square. There is no silicification and no mineral. The Bullfrog claims cover an immense outcrop that can be seen for miles, and are only 3 miles from a spring that has been frequented by prospectors for 30 years or more. The white quartz lies like a crescent around a small dome-shaped hill, following the contact which dips 30° N. 60° W. into the hill. This quartz has a maximum thickness of nearly 100 ft., and is generally massive, though sometimes there are large slightly amethystine-tinted crystals with a conchoidal base. A later cracking has occurred, with a flow of water depositing copper sulphides and the precious metals. Wherever a green stain occurs visible gold can usually be found.

The other properties of this district are entirely different, resembling somewhat those of Goldfield. There has been a nearly vertical fissuring, followed by a flow of water heavily charged with silica, filling the fissures and soaking into the country-rock. One can find all gradations from pure quartz to slightly silicified country-rock. These are the so-called rhyolite

dikes. The country-rock itself had a slight mineralization, which this silicification did not increase. A secondary and much less extensive cracking and inflow of siliceous waters occurred, which deposited the gold. The veins are nearly vertical, strike N. 10° to 30° E., are sometimes thin seams, sometimes several feet in thickness, and again wide zones of stockwerk. The better-formed are calcareous and slightly stained with manganese. There are many of these veins, and they were easily found; but to discover ore-shoots in them is quite a different matter. The amount of work done in both this district and Goldfield during the past winter has been remarkably small.

An interesting and, to the prospector, a very important phenomenon is the covering of these veins. The older tuffs are mineralized and the more recent (upper) are not; the older are basic, while in the upper there is a flow of rhyolite. At Tonopah it is quite noticeable that the rhyolite is more recent than the "mineralized porphyry." The regularity of the eruptions and the exposure of the edges of the often highly colored strata make this an ideal place to study this phenomenon.

V. Funeral Range.

The Grapevine and Funeral ranges are practically the same. Old-timers do not agree as to the dividing line. With the mountains south, they form the eastern boundary of Death valley. There has been much searching in this range for the lost Breyfogle mine, one of the romances of the desert. Except in one place, both ranges are poor prospecting-ground. The general formation is quartzite and limestone overlain by immense deposits of recent conglomerate. In the north end of the Funeral range there is a development of green shales, identified elsewhere as Cambrian. These shales usually carry white glassy quartz, which is rarely mineralized; but in this case there are ore-shoots, carrying sufficient gold to make the district attractive if wood and water were more available. One property has recently been thoroughly developed under bond, in this district.

To the south, the range ends abruptly, near where the wagon-road from Ash Meadows to Furnace creek crosses. The division along the dry water-courses followed by this road is re-

markable; they have large boulders of quartzite and limestone north, and on the other side black and brown lava. South, the topography is broken and mountainous, but shows no distinct range. Green mountain, still farther SW., is another field for prospectors. One small stamp-mill is running at the foot of this mountain on the Death valley side. Still farther south, the drainage of the Amargosa cuts through into Death valley.

VI. Goldfield.

The tuffs (andesite) of this district probably lie upon limestone, elsewhere identified as Cambrian. On the southwest side of Columbia mountain is an outcrop of dark limestone, evidently overflowed by eruptives. In the Tonopah Club claim the ore is a siliceous sedimentary that has the appearance of limestone, through which gold-bearing solutions have percolated, leaving enough of the precious metal to make it possible to sort out some shipping-ore.

The country-rock shows more mineralization than in the southern district, and has been much more disturbed since the mineral deposition. There are three large intrusions of alaskite, and a great many dikes of a green rock, probably decomposed trap, which is closely associated with the original andesite. Standing upon one of the intrusions, Vindicator mountain, one can see, both from the workings and the color of the country-rock, that the mineralized area takes the form of a ring. It is yet a question whether the central part will prove of value or not. Outside, the field is completely surrounded by more recent tuffs, overflowed on the west side by dark basaltic lava, which forms a mesa some 4 miles square between Goldfield town and Montezuma mountain.

The topography presents simply hills, with higher hills or low mountains around them, except to the north, which opens out to the San Antonio desert. Montezuma mountain (altitude 8,000 ft.) is 8 miles west. The old Montezuma lead-silver district is on the west slope of this mountain. The castings in the 10-stamp mill and 36-in. water-jacket furnace bear the date of 1886. At Lida, a boiler-front recently re-set was cast in 1866.

Very prominent geological features are the reefs of silicified country-rock, usually called rhyolite dikes, found all over the

district, but more numerous to the south and east. They are sometimes very large (50 ft.), but generally about 10 ft. thick; and they extend in all directions without regularity, frequently crossing each other. They are sometimes 2,000 or 3,000 ft. long, and again but a few feet, and exhibit all grades of silicification. On the surface they are hard and flinty; but underground, away from the weather, the rock, although harder than the andesite, which is quite soft, is not bad for drilling. The phenomenon of blackening, so noticeable in the desert and recently described by Prof. Blake,[3] is very apparent.

The gold was deposited by successive flows of water in or near these reefs, as was the case with the dikes at Victor, Colo., but there are exceptions, as in the Velvet and Tonopah Club claims. Again, as at Cripple Creek, one often hears that the country-rock shows value, and not the veins. At the surface the gold is very free, and is fine in both grain and grade. Below the zone of oxidation, the ores are not yet thoroughly understood. There is probably considerable difference in the different properties. It is now certain that tellurides are present; but the principal accompanying mineral is, as usual, pyrite. Although there is no copper-stain near the surface, there is considerable gray copper at depth. There is rather more antimony than arsenic, but both are present.

It is too early to say what will be the solution of the metallurgical problem. The low-grade ores, of which so little is known now, may be more simple in their composition. The Combination Mines Co. has provided amalgamation-plates, concentrators and a cyanide plant.

It is not at all easy to find the small rich streaks and lenses of ore that have given Goldfield its celebrity, and there is as yet no incentive to develop the low-grade or milling-ores, which will later be the important ones. The charges for freight and treatment are now $32 per ton.

The Jumbo was sampled many times and "turned down." It was bonded and a shaft was sunk, with no results. On the Quartzite, a shaft was sunk and trenching done, and the property was given up. On November 10, 1902, H. Stimler and W. A. Marsh camped at Rabbit Spring, where Goldfield now

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[3] *Trans.*, xxxv., 371 (1905).

gets its water, and soon after made locations in the Sandstorm section. In the spring of 1904, the Sandstorm and Kendall claims were carefully explored, with no results. In the Jumbo, gold was found by panning the soft rich country-rock close to the reef, which was rich; but the vein was found later in the reef, through which it takes a zigzag course.

After the incorporation of the Jumbo Mining Co., leases were let for the last seven months of 1904 according to the custom of the district, in blocks 200 ft. long by the width of the claim, royalties being set at 25 per cent. of the gross yield. These leases yielded over $1,000,000. The best piece of ground, 200 ft. long and 200 ft. deep, yielded in round figures 2,000 tons of $350 ore and 3,000 tons of $50 ore. The Quartzite and the Sandstorm are now among the active shippers. One lot of 14.5 tons from the Kendall claim of the Sandstorm group, yielded in the mill $45,785; and tailings valued at $1,000 (in another report, $300) per ton.

The Combination Mines Co. is the only corporation in the district that is carefully preparing for regular future production. Good buildings are under erection, a pipe-line has been laid 10 miles to a spring, and a well-built mill has been completed. The principal vein is parallel to that of the Jumbo. Just to the NW., on a cross-vein, is the January, which has a dump of several thousand tons of milling-ore awaiting treatment. The Florence mine, one of the best, is on another cross-vein E. of the Combination. This group of four mines is the most important in the district.

Four miles north of Goldfield town is Diamondfield, about 1 mile N. of which occurs another group of promising mines from which some shipments have been made. One of these is the Black Butte, a prominent topographical feature. On this property has been developed ore of probably the lowest grade ($20) mined in the district. A short distance north, on the slope of the butte, is the Quartzite "fraction," one of the most promising properties now shipping ore. Half a mile N. is the Vernal, which has also shipped some ore. To the northwest are two very strong quartz-reefs, in which very limited prospecting has not yet developed any important ore-bodies.

The explorations near the Sandstorm, 3 miles W. of Diamondfield, and a little further from Goldfield, have not been

specially fortunate, except in the Tonopah Club, which is in the low ground between Diamondfield and the Sandstorm.

Northeast of Goldfield town some important discoveries have been made; and ore has been shipped, notably from the St. Ives, a claim covering a very prominent reef near the Jumbo, and from the Cimerone. The latter was found during the summer of 1904, and the finder literally camped on it, putting his tent and bed over the rich place, until he had succeeded in buying the fractional claim from the original locator. Then he made the discovery public, and, in a few weeks, sold out, it is said, for $60,000 cash.

In March, 1905, the town of Goldfield, lying between the Combination mine and the mesa at the foot of Montezuma mountain, had an estimated population of 10,000, and Columbia, practically an extension 1 mile NW., had 2,000, and Diamondfield about 500. Goldfield and Columbia are supplied with excellent water by two 2-in. pipe-lines. The water comes from under the mesa immediately west. The Combination Mines Co. has a pipe-line from the Warm Springs. Twenty miles west of Goldfield is the Silver Peak marsh, where there is an open lake.

Both at Bullfrog and at Goldfield the situation is somewhat discouraging. In March last, scarcely 50 men were working at Bullfrog, and about 200 at Goldfield. While actual development was thus neglected, the industry of transferring to new corporations groups of claims, good, bad and indifferent, and of selling the stock of such corporations, was active. All American mining districts have passed, and will doubtless hereafter pass, through such a period.

VII. Tonopah.

The veins in this district are much stronger and more condensed, and possess the regularity of silver-veins, which they should be deemed to be, since the values are about two-thirds in that metal. The railroad has been in Tonopah only a year, and has been overwhelmed with freight for the newer districts. One small mill, owned locally, is operating. The owners of the developed properties do not seem to think that the time is yet ripe for large reduction-plants.

About 1,500 tons of high-grade ore is shipped weekly to the

smelters, and, incidentally, an enormous amount of milling-ore is blocked out. It is said that one company has $35,000,000 in "positive ore." In one of the mines 60 ft. thickness of $28 ore is reported. The maximum depth reached is 1,000 feet. This district has therefore long passed its doubtful stage.

The country-rock is andesite, so overflowed by more recent volcanics that very little of it is exposed. The explorations of the past year have gone through this overflow and found ore by drifting in the "mineralized porphyry" below.

The development of milling, always an expensive and interesting problem, will be unusually so in these three districts. Their ores, though different, are all typically milling-ores, consisting of quartz with very little base metal. Should smelting be necessary, there are several flux-producing districts, now idle, which could be drawn upon.

VIII. Borax-Deposits.

South of the Funeral range, in the region drained by Furnace creek and on the Amargosa side of the mountain, is a large development of recent sedimentaries—shales, clays, sandstones, and thin sheets of gypsum. There are a few intrusions of andesite, and a heavy flow of black and brown basaltic lava. In these sedimentaries immense deposits of calcium borate occur, conformable with the strata. The most common mineral is colemanite. As one might expect from an element possessing the peculiar solubilities of borax, there are many combinations of boric acid, lime and soda under various mineralogical names. There is some question as to the origin of the deposits. The Pacific Coast Borax Co. first obtained title under the placer-law, but now favors lode-locations. That company has the region pretty well "corralled" for borax. The rocks are highly colored. The country is bare of vegetation, and water is scarce.

The borax-deposits are remarkable in size and purity. In one place there is an outcrop of calcium borate 30 ft. thick. At the Lila C. mine on the Amargosa side of the range, 35 miles from and in sight of Ash Meadows, is a deposit, from 3 to 17 ft. thick, dipping about 45° E., and explored on the surface for a mile. The underground workings are in the vein (?)

for a quarter of a mile. This is the property that will bring a railroad to this section.

IX. Death Valley.

Much has been written about this valley, and a strange amount of romance is attached to it. The prospector could easily find a more promising field and a less expensive place to work a mine in. It is a long narrow valley, very deep and surrounded by high mountains. The Panamint and Sentinel peaks reach an extreme elevation of 11,000 ft., while Death valley, hardly 10 miles west, is two or three hundred feet below sea-level. A similar difference of elevation occurs 80 miles NW., between the summit of Mount Whitney and Owens Lake valley; only, the Owens Lake valley is not below sea-level.

Under one general name there are three connecting valleys: Death valley proper, Lost valley and Mesquite valley. The upper end of the latter is only about 30 miles from Goldfield. Instead of being the horrible region usually reported, it is the best of the desert valleys. Lying so low and being shut in by surrounding mountains, it is hot in summer; but the winter climate leaves little to be desired, particularly by those who require a dry atmosphere. But a few miles west there is an elevation in the Panamints, where in the shade of the pines the traveler can be comfortable, and look into the sweltering valley below, while a battery of abandoned charcoal-kilns might make him fancy he was summering near Lake Superior.

The name Death valley comes from the loss of an emigrant train in the lower end of Lost valley. The party was on the way from Salt Lake to southern California, and, becoming exhausted, stopped to rest in what appeared to be a meadow. The salt and wire grass is not nutritious; the water is saline and carries enough sulphates to disarrange promptly the human digestive system. The spot, where it is said about 50 people perished, has been dug over for buried treasure; and last winter many of the pits were in brackish water. Some prospectors, also, have lost their lives in the south end.

At the mouth of Furnace creek, the Pacific Coast Borax Co. maintains a ranch, having 200 acres in alfalfa and wheat. Twice a month a 16-mule team arrives from Dagget, 160 miles away on the Santa Fé railroad. Three miles from the ranch are the old Coleman borax-works.

Furnace creek and several streams south, usually dry, bring down borax in solution. For a few miles in width and a length of about 30 miles, the lower part of this valley looks like a mud-flat with the tide out. These places are locally called marshes, although they have no vegetation. The borax is here a double borate of lime and soda, commonly called "cotton-ball."

A peculiar phenomenon, better seen in this than in the other valleys, is the "self-rising ground." The soil carries a large proportion of soluble salts, sodium carbonate, sodium chloride, and various sulphates and borates. Evaporation is excessive; the subsoil is moist, being constantly supplied by springs; and capillary attraction brings the salts to the surface. This does not go on evenly, but forms hummocks, sometimes 2 or 3 ft. high, hard on the surface and soft beneath, making a bad and sometimes dangerous ground to walk across. The same natural forces have caused the surface-enrichment of some mineral-veins in the desert, particularly veins of copper-ore.

The atmosphere in the valley is remarkably clear and possesses the resonance so noticeable above timber-line. There is no truth in the story about birds and animals dying in attempting to cross the valley. The Indians and the Borax Co. keep several hundred head of cattle and some mules and horses about the mouth of Furnace creek; and rabbits, quail and other small game can be found.

X. Panamint Range.

This range is unusually high and precipitous, starting at sea-level instead of several thousand feet above, as most other high ranges do, and having no foot-hills. The rocks are green slates, mica schist, quartzite and limestone. On the west side is an intrusive granite which has tilted the whole formation, the larger part of it to the east. The green slates carry fine-looking quartz; but so far it has not been found to carry values. On top of the range are large areas of recent conglomerate and basaltic lava.

Panamint City once had 6,000 people, but is now reduced to a few old-timers, who say it will soon start again and make "the greatest camp on earth." The ores are of silver—refrac-

tory and heavy, with a barytic gangue. The railroad is 90 miles away.

On the west side of the mountain, every gulch for 46 miles has some sort of a mining equipment, usually a small stamp-mill, idle, and owned by some Los Angeles company. The first question asked of strangers, is "Are you from Los Angeles?" It is best to answer, "No!"

XI. Lead-Mines.

South of the Amargosa there are some lead-ores and two large deposits of iron-ore. They are too far from a railroad to have anything but speculative value.

The mountains between the Panamints and the Sierra Nevada were once the scene of great activity in lead-silver mining. Old roads constructed at great expense, smelting-plants at the mines, and charcoal-kilns many miles away in the timbered mountains, are mute evidences of this former activity.

Cerro Gordo, Darwin and Modoc produced between 1870 and 1880 approximately $25,000,000. The first-named was the heaviest producer, furnishing the largest quantity and lowest grade of ore, while the last produced the smallest tonnage and the highest grade. They are all at or near the contact of granite and limestone. Unlike the deposits at Monarch, Colo., along a similar contact, these ore-bodies are from a few feet to 300 ft. away from the contact, in cracks and crevices of the limestone.

At Darwin there is an anticline about 6 miles long, the west side of which dips about 45° and the east side more steeply. The latter carries some copper, while the former shows none. The oxidized surface-ores contain roughly 1 oz. of silver to 1 per cent. of lead, and the galena ores 2 oz. of silver to 1 per cent. of lead. From the old books left at some of the works, it would appear that the average ore contained from 40 to 60 per cent. of lead. The gangue carries iron and lime, with some silica. There are four smelting-plants near by, one of which is in good condition; and 20 persons are still living in the town.

The popular report is that the ores became too poor to work at a depth of 800 feet. While these mines were working, Mojave, Cal., was the nearest railroad point, and wood and

charcoal had to be hauled long distances. From the charcoal-kilns in the Panamints to Darwin is 50 miles. The altitude of the kilns is 8,000, that of Darwin 6,000, and that of the intervening Panamint valley 1,100 ft. above tide. The price of fodder must have been high. The nearest farms are now 50 miles away.

Cerro Gordo is 7 miles from, and 3,500 ft. above, Keeler, the terminus of the Carson and Colorado railroad, 334 miles from Reno, and, by wagon-road, 120 miles from Mojave. Here are located the soda-works of the Inyo County Development Co. With diminished treatment-charges at custom-smelters, lower rates of freight, and the flood of siliceous ores now going on to the market, it would not be surprising if these lead-ores, with their useful fluxing character, should be again mined with profit.

XII. Owens Lake Valley.

Owens Lake valley, about 75 miles long and 20 wide, is drained by Owens river, which flows into the lake of the same name, about 18 by 12 miles in area. The water of the lake is a nearly saturated solution of sodium carbonate and common salt, with a little sulphates and borax. There is no verdure around the edges.

The river is fed by streams from the west, having their origin in the Sierra Nevada, a very high, snowy and well-timbered range. All the older settlements are on these streams, but the railroad follows the east side of the valley. A large ditch has been carried by an irrigation company down the east side of the valley to within 12 miles of Keeler; and the new-comers are settling along this ditch.

While the valley is fertile and well-watered, particularly at the north end, the farmers have not as a rule been prosperous, because the market was too distant. Now a sudden change has come. From Laws station, opposite Bishop, in the upper end of the valley, to Tonopah is 113 miles by rail. It is fortunate, both for the farmers and the miners, that there should be an agricultural region so near.

All the streams coming from the Sierra Nevada furnish opportunities for developing water-power. Already a company is preparing to generate and transmit electric power from Bishop creek to Tonopah and Goldfield.

All the grains and fruits of the temperate zone are raised here. The apples, peaches, pears and certain varieties of grapes, are better than those raised on the Coast-side of the Sierra.

XIII. Smaller Mines Tributary to Owens Lake Valley.

In the south end of the Argus range and in the Coso mountains are many veins, usually only a few feet in width, of white quartz in granite, occasionally carrying gold, low in value, and in spots rather than in regular ore-bodies. Most mining men dislike these conditions. The Congress mine in Arizona is the only large and successful enterprise working this class of quartz.

The Beveridge and Lee districts, NW. of Darwin, have argentiferous ores. At the latter place there was formerly a stamp-mill, which, judging from the amount of tailings, did not run long.

The Ubaheba district, lying between the Saline and Butte valleys, is a large undeveloped area of low-grade copper-ores, in contact-deposits between limestone and some acid eruptive. These valleys are very deep, and would be a continuation of the Panamint valley but for an east and west mountain that looks like an enormous dam. The west end is granite; but the larger portion of the mountain is recent conglomerate overflowed by basalt.

In the mountains facing Owens Lake valley on the east side, both in the granite and the chloritic slates, quartzites and limestones, which the intrusive has thrown on edge, there are numerous veins of white quartz, carrying occasional gold-values.

Between the stations of Alvord and Citrus, 3 or 4 miles from the railroad at the foot of the mountain, and well located for economic reasons, are a new stamp-mill and out-buildings, now idle.

Farther up the valley, at Poleta, there is a plant running.

XIV. From Owens Lake Valley to Goldfield.

From Alvord station, 5 miles from Big Pine, 20 miles from Bishop, and 54 miles by rail from Keeler, there is a wagon-road, 61 miles to Lida, and 96 miles to Goldfield. This road is now much used by freighters and farmers hauling produce to Lida, Goldfield and Tonopah. Three ranges of mountains

and two valleys are crossed. Between the Deep Spring and Fish Lake valleys there is a gold-copper exploration near the road. In the mountain east of Fish lake are two old mills.

About Lida, and between Lida and Montezuma mountain, the formation is light-green slates overlain nonconformably by limestone. In the slates are dikes of porphyry and rhyolite, and many quartz-veins. At Lida the veins are exposed on the surface and show remarkable persistence in length; but when worked, 30 years ago, they were found to lose their values at a depth of from 200 to 300 feet. The ore is quartz with little galena and zinc blende. The principal values are in silver.

From Lida east to the Kawich range the rocks are all volcanic, from rhyolite to basalt inclusive, but rarely are there any of the earlier tuffs or andesites.

Gold Center is a small area, similar to Goldfield, but the ores are not particularly high-grade or continuous. The soft aluminous country-rock seems to have moved too much after the ore was deposited.

Quartz Mountain, 24 miles south, is of rhyolite, with veins similar to those of the Bullfrog district.

It is remarkable that so many mines have been found of late years in the volcanic tuffs, now generally known as andesite. A very large portion of them carry gold. One cannot but wonder if there are not more. This is not a formation which prospectors have liked until lately; and as yet it has been but imperfectly studied. The fact that the mineral-bearing tuffs are basic, and are overlain by the acid rhyolite, is perhaps significant. Cripple Creek has a rhyolite mountain in which much money has been spent without satisfactory results. The nature of the veins, too, is new. They may often be called freaks. Mineralizations of country-rock are to be expected rather than "text-book" veins, such as used to be sought for.

An Old Specimen of American Spiegeleisen.

BY FRANK FIRMSTONE, EASTON, PA.

(Bethlehem Meeting, February, 1906.)

THE piece of spiegeleisen, the analysis of which is given below, was collected by my father, together with various other specimens, while he was manager of the Glendon Iron Works. It bears a label stating that it was taken from a building in the town of Newton, New Jersey, erected in 1765 and torn down in 1863. The building belonged, in 1863, to T. N. McCarter, Esq. The label is in the hand-writing of Mr. McCarter, now deceased, well known as a member of the bar of northern New Jersey. The specimen was, no doubt, given by him to my father, with whom he had business in 1863 and 1864. It is now in the metallurgical collection of Lehigh University.

I know nothing further as to the history of the specimen, but there is no reasonable doubt that it was made from ore from the Andover mine, at the old charcoal-furnace which once stood at the village of Andover, about 5 miles south of Newton, on the Sussex railroad. It was probably broken from one end of a bearer over an open fire-place, is about 6 in. wide by 2.5 in. thick, has been cast in open sand and shows the characteristic fracture of spiegeleisen rather low in manganese, as shown in Table I. One edge has been white-washed.

TABLE I.—*Analyses of Spiegeleisen Made from Andover Ore.*

	I. Per Cent.	II. Per Cent
Graphite,	0.101	0.005
Combined carbon,	4.529	4.212
Silicon,	0.037	0.294
Manganese,	5.750	3.750
Sulphur,	0.003	0.031
Phosphorus,	0.060	0.072
Iron,	89.440	91.570
	99.920	99.934

I. Spiegeleisen made at Andover (charcoal) furnace not later than 1765.

II. Spiegeleisen made from Andover ore at Phillipsburg, N. J., anthracite-furnace, 185–.

Both analyses were made by Dr. P. W. Shimer.

There is nothing to be seen to-day of the old furnace at the reputed site in the village of Andover; but there is a large cinder-heap there, and I have no doubt, therefore, that this is really where it stood. This cinder-pile is on a small stream, the outlet of a lake south of Andover, and a little east of the railroad. The water runs into the Pequest. Just below the cinder-pile is an old stone grist-mill, still in use. The date 1761 is cut on a stone on the northwest corner, about 18 in. under the eaves. This mill, no doubt, was built about the same time as the furnace, a mill being at that period an indispensable adjunct to an iron-works.

Mr. W. R. Ayers, of Andover, who pointed out this inscription to me, and kindly gave me much other information, has in his possession a pig of iron about 2 ft. 10 in. long with the brand "Andover" on it in very well formed letters. It was taken from a house belonging to him, in which it formed a lintel over an open fire-place. The house is said to have been built before the Revolution. This pig, judging by the face, is not spiegeleisen.

Almost all the cinder to be seen on the old dump is gray, and indicates very regular furnace-work. Table II. gives analyses of two pieces which I took from the dump on my first visit to Andover.

TABLE II.—*Analyses of Cinder from Old Andover Charcoal-Furnace.*

	No. 1. Per Cent.	No. 2. Per Cent
SiO_2,	45.29	36.57
Al_2O_3,	2.67	5.49
FeO,	0.55	1.44
MnO,	7.20	4.43
CaO,	34.74	41.17
MgO,	6.53	8.77
K_2O,	0.49	0.33
Na_2O,	0.11	0.09
SO_3,	0.137 (S, 0.055)	0.191 (S, 0.077)
CO_2,	1.59	0.74
Combined water,	0.49	0.40
	99.797	99.621

The last three constituents are, doubtless, due to long weathering.

Sample No. 1 is highly crystalline in fracture; No. 2 is compact and shows no signs of crystallization. Both are of rather

unusual composition for charcoal cinders, although No. 1 is not very different from a cinder from Långshyttan (Sweden) given by Åkerman.[1]

I think the spiegeleisen made at the old Andover furnace must have been the first produced on this side of the Atlantic.

The Andover furnace has additional historical interest in that, as appears in Swank's account of it[2] and the documents there referred to, it was taken into possession by the government in 1778, and run to furnish iron for steel for the War Department, this being the only iron which would "with certainty answer the purpose of making steel."

If the accounts of the Continental War Department have been preserved and are accessible, they would furnish amusing, and perhaps important, details of methods, costs, etc.

The Andover mine, about a mile north of the furnace-site, is described in Kitchell's *Annual Report on the Geological Survey of the State of New Jersey for the Year* 1855 (Trenton, 1856), pp. 149 *et seq.*; and there are further notes in Lesley's *Iron Manufacturer's Guide*, p. 427. The description in Kitchell's report is reprinted in Prof. Geo. H. Cook's account of the mine,[3] with some historical details taken from a letter from Hon. A. S. Hewitt.

In 1848 and 1849 the mine was re-opened by the Trenton Iron Co. (Messrs. Cooper & Hewitt); and much iron was made from the ore in the '50's at the furnaces of that company at Phillipsburg, N. J., at first under the management of Dr. G. G. Palmer, who built the works, and afterwards under his immediate successor, our fellow-member, Mr. J. C. Kent. Much of this iron was spiegeleisen, and large quantities were puddled at the mills of the same company in Trenton, and, no doubt, contributed to the good quality of the "Trenton" rails, which were famous in their day. Through Mr. S. B. Patterson I have procured a small piece of the spiegeleisen made at Phillipsburg, and the analysis (Sample II.) is given above. Unfortunately, the exact date of manufacture is not known.

Mr. Kent informs me that the furnaces worked very well on this ore, and that over 300 tons of iron per furnace per week

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[1] *Stahl und Eisen*, No. 6, 1886, p. 388, Table VII.

[2] *Iron in All Ages*, 2d ed., p. 153 (1892).

[3] *Geology of New Jersey*, 1868, p. 640 *et seq.*

were sometimes made—which for that day, and with anthracite for fuel, was an enormous output. He also tells me that Mr. Peter Cooper, knowing that the iron had been used for steel in Revolutionary times, had steel made from some of the first iron produced at Phillipsburg and Trenton, and that this steel was worked up into cutlery.

The mine was practically exhausted before the successful introduction of the Bessemer process into this country, otherwise spiegeleisen of such good quality must have played an important part in the early history of that manufacture here.

In 1868 Messrs. Cooper & Hewitt sold the Phillipsburg furnaces, the Andover mine-tract, and various other mineral properties and leases in New Jersey, to a joint stock-company, which took the name of Andover Iron Co. In 1901 this company and its belongings passed, by purchase of the stock, into the hands of Mr. Joseph Wharton.

Notes on the Gayley Dry-Air Blast-Process.

BY C. A. MEISSNER, NEW YORK.

(Bethlehem Meeting, February, 1906.)

THE following is a further discussion of the paper of James Gayley, "The Application of Dry-Air Blast to the Manufacture of Iron" (*Trans.*, xxxv., 746), with special reference to his supplementary paper (*Bi-Monthly Bulletin*, No. 4, July, 1905, pp. 797 to 819), and to the paper of J. E. Johnson, Jr., "Notes on the Physical Action of the Blast Furnace" (*Bi-Monthly Bulletin*, No. 5, September, 1905, pp. 1111 to 1146).

Mr. Gayley's paper has been discussed also by E. Windsor Richards, R. W. Raymond, E. H. Saniter, W. J. Foster, H. M. Howe, V. Pendred, B. H. Thwaite, James Gayley and Alexander Pourcel (*Trans.*, xxxv., pp. 1022 to 1042), Joseph W. Richards (*Bi-Monthly Bulletin*, No. 4, July, 1905, pp. 703 to 718); T. W. Robinson (*Bi-Monthly Bulletin*, No. 4, July, 1905, pp. 788 to 796); C. B. Dudley, R. W. Raymond, J. E. Johnson, Jr., W. F. Mattes, James Gayley, David Baker, John Birkinbine and F. E. Bachman (*Bi-Monthly Bulletin*, No. 5, Septem-

ber, 1905, pp. 1147 to 1152); E. de Mille Campbell (*Bi-Monthly Bulletin*, No. 1, January, 1906, pp. 25 to 50); John Birkinbine (*Bi-Monthly Bulletin*, No. 1, January, 1906, pp. 137, 138).

At the Washington meeting of the Institute, May, 1905, a very instructive and interesting paper was presented by Mr. J. E. Johnson, Jr., entitled "Notes on the Physical Action of the Blast Furnace,"[1] in which reference is made to Mr. Gayley's dry-blast process. A few of Mr. Johnson's conclusions, as summarized in the last pages of his paper, do not appear to me to be in complete accordance with the results obtained at the Isabella furnaces; and I therefore present the following notes, which may throw further light on this subject:

Mr. Gayley has called attention to the fact that the moisture in the air is an extremely variable and uncertain factor; and that, while every possible effort has been made to control the composition of the burden of raw materials within 10 per cent. of variation, yet there has been no such control of the composition of the blast, the actual weight of which is 1.43 to 1 of the weight of the solid charge, and which may vary daily from 20 to 100 per cent. in its content of moisture and in its temperature. Those who have studied the subject know that these conditions exist wherever iron is made. The gist of Mr. Gayley's labors in this direction is the endeavor to secure a uniform composition of the air-supply (which, after all, is the largest and most important factor in the manufacture of iron), as the furnaceman does for the mixture of fuel, ore and flux constituting the solid charge.

Various suggestions to bring about this result have been made by many writers; and, in some cases, it has been argued that the reported results of Mr. Gayley's dry-air blast, showing a gain of from 10 to 20 per cent. in the output of pig-iron, and a saving of from 10 to 20 per cent. in coke-consumption, were due to natural air-conditions existing at the time of his experiments. It has also been pointed out that these reported economies were obtained in the winter months. This criticism, however plausible, was based on insufficient data; Mr. Gayley's first figures having covered, in fact, but a comparatively short period, and that largely in the winter months. But, according to later experience, covering summer months as well,

[1] *Trans.*, xxxvi., 454 to 488 (1906).

all the dry-air blast periods indicate a similar tendency to continuous higher pig-iron production and a lower coke consumption. This is shown by additional data given below.

A comparison of the natural-blast records of Furnace No. 3 for the winter months, February and March, 1903, with similar previous records, showed that the coke-consumption in these winter months (2,225 lb. per ton of pig-iron produced) did not differ materially from that of the summer months. In 1904, like results were obtained; the fuel consumption (2,297 lb. per ton of pig-iron produced) being practically constant throughout the year; in fact, the monthly records of both furnaces during the natural-air periods preceding the dry-blast periods showed as uniform and steady operations as is possible to get under such conditions. In 1905, however, the records show that the lowest coke-consumption, averaging about 1,800 lb. per ton of pig-iron produced, was obtained during the three winter months of the dry-air period of Furnace No. 3, showing that the gain from the use of dry-blast is by no means restricted to the summer months. In the winter months, given above, Furnace No. 3, running on mill-iron, saved at least 20 per cent. in coke-consumption, as compared with that required by natural air, under otherwise similar conditions, in the winter months of the two preceding years.

It has also been said that in very cold, dry periods, an increase in pig-iron production, and a decrease in coke-consumption, equal to those reported by Mr. Gayley, have been obtained. It is to be regretted that these statements were not accompanied with tabulated data, permitting critical comparisons. Such data should show the length of time covered, and, especially, whether the 20 per cent. increase of pig-iron production and the 20 per cent. decrease in coke-consumption were obtained simultaneously.

I have in mind here Mr. Walter Crooks' graphic statement[2] showing a gain in pig-iron production and a decrease in coke-consumption during winter months over summer months for two years; yet even these figures show but a 6 per cent. increase in pig-iron produced and 9 per cent. decrease in the quantity of coke consumed. No data are given showing the actual

[2] Discussion, *Journal of the Iron and Steel Institute*, vol. lxvii., No. 1, pp. 267, 268 (1905).

production of pig-iron, the consumption of coke or the temperature of the blast.

Among the records which I have been able to study (those of about 90 furnaces for from four to six years back), there are none which show that any such percentage of iron-increase and coke-decrease either took place simultaneously under natural dry atmospheric conditions, or even covered separately any long period. They often show a distinct tendency to an increase in the pig-iron production during such natural dry-air conditions; but with a few exceptions, practically stationary or but slightly varying coke-consumption. In the early part of November, 1905, when the atmospheric conditions gave an almost dry air, one of the Isabella furnaces, running with natural blast, increased its production from 375 to 430 tons per day; yet during all this time the coke-consumption remained practically stationary, the increased iron-production being due to the temporary ability to blow harder, which meant an increase in the number of revolutions. Moreover, this lasted only eight or ten days; so that it presented no parallel to Mr. Gayley's results, which were obtained with a decreased number of revolutions and an immediate and simultaneous increase in the production of iron and a decrease in the coke-consumption.

In short, the economies effected by the use of the dry-air blast are now shown to be uniform and continuous for periods of months together, and to be practically unaffected by stoppages, repairs, or other disturbances. These economies have been achieved with two furnaces that had run for more than two and three years, had produced respectively more than 300,000 and nearly 400,000 tons of pig-iron upon their existing linings, had been repeatedly banked, and were more or less damaged. I think this record cannot be matched from experience with natural-air blast, and is, therefore, conclusive as to the value of Mr. Gayley's invention.

Another point brought out by some of the arguments on the subject, and one which is apparently carrying great weight in many minds, is the one of the temperature of the blast.

It has been stated frequently that an increase in blast-temperature alone would accomplish the same results as those obtained by Mr. Gayley. The necessary increase of temperature to do this has been variously estimated at from 150° to 400° F.,

and the estimates of the amount of coke saved by every 100° of increased blast-temperature vary considerably.

Table I., giving the average blast-temperature during the natural-blast and dry-blast periods of the two furnaces named, shows a difference of average of temperature between the first natural-air period and the dry-blast period of Furnace No. 1, of about 50° F. only, while in the second natural-air period, following that of the dry-blast, the average blast-temperature was actually about 10° F. higher. For Furnace No. 3, the natural-air temperature was only 45° below that of the dry-blast.

According to Fig. 2 of Mr. Johnson's paper,[3] air at 800° F., containing 5 grains of moisture per cu. ft., would yield about 1,200 B.t.u., and air at 850° F., with 2 grains of moisture per cu. ft., would yield about 1,450 B.t.u. of "available heat" per lb. of fuel consumed. The latter figure fairly represents dry-blast conditions, and (assuming the average coke-consumption with natural-blast as 2,275 lb. per ton of pig-iron produced) indicates a coke-consumption, for dry-blast, of 1,875 lb., which closely accords with practice.

We may proceed to inquire what would be the effect of dry-blast combined with higher temperature—say, 1,200° F., which is 350° F. more than that of the blast at the Isabella furnaces during the dry-blast period in which the coke-consumption was reduced to 1,850 lb. According to Mr. Johnson's diagram, this increase from 850° F. to 1,200° F., on a blast containing 2 grains of moisture per cu. ft., should increase the "available heat" from 1,450 to about 1,850 B.t.u. Theoretically, this would reduce the coke-consumption of 1,875 lb. for the former temperature, to about 1,470 lb. for the latter. But, if we apply the practical factor of 50 lb. less coke per ton of pig-iron for every additional 100° F. of blast-temperature, we shall have, in the case stated, a saving of 175 lb. of coke per 350° F. of additional temperature; and consequently, for blast at 1,200° F. containing 2 grains of moisture per cu. ft., a coke-consumption of 1,700 lb. per ton of pig-iron produced. This appears to be a reasonable expectation, according to our present knowledge of the effects of the dry-blast.

Further study of the monthly blast-temperatures showed

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[3] *Trans.*, xxxvi., 475 (1906).

TABLE I.—*Comparison of Natural Air and Dry-Blast*

Period.	No. of Days in Period.	Kind of Iron.	Av. Tons Pig Per Day.	Av. Coke Consumption per Ton Pig.	Temp. Degrees F.		Engine Rev. Per Min.	Grains of Moisture.			
								Atmosphere		Dry Blast.	
					Blast.	Gases		Day.	Night	Day.	Night
urnace No 1.											
First natural-air period...... From July to Oct. 15, 1903. Banked three months. Jan. to Aug., 1904.	314	Basic. Besse'r.	357	2,275	807	516	113	3.93	3.92	. .....	....
Dry-blast period .... From Aug., 1904, to Jan., 1905, including also Aug., 1905	164	Basic. Besse'r.	436	1,821	854	397	96	3.50	3.56	1 33	1.24
Second natural-air period. ... From Jan to Aug., 1905.	205	Basic. Besse'r	383	2,256	863	501	108	3.44	3.60	....	....
urnace No. 3											
First natural-air period......... From Jan. to Sept., 1903. Banked four months. Feb. to May, 1904. Banked three months. Sep. to Nov., 1904.	478	Basic. Besse'r.	370	2,258	775	....	108	3.22	3.35	....	....
Partial dry-blast period.. . . Natural air December, 1904 Dec., 1904, to Jan. 15, 1905.	21	Basic	399	2,307	....	.. ..	....	... .	. . ..	....	....
*One engine dry-blast Dec [illegible]	10	Basic.	453	2,204	785	....	104	1.41	1.47	....	....
[illegible]	10	Basic.	409	2,329	... ...	.. ... .	..... .	......	....	....	....
[illegible]	4	Basic.	425	1,930	680	....	111	1.46	1.46	....	....
[illegible] Jan.	17	Basic.	432	1,824	... .	....	....	... .	. ...	....	....
Dry-blast period .... From Jan. to Aug., 1905.	203	Besse'r.	400	1,961	820	448	93	3 44	3.60	1.26	1.17

* Three engines on each furnace.

that, in many cases, the monthly averages for natural air were higher than for dry-blast, and yet that neither coke-consumption nor product of iron differed materially in those particular months from the corresponding figures for other months of natural-air blast, characterized by 100° F. or 150° F. lower blast-temperature. In other words, the effects of the blast-temperature, within these limits, were counteracted by other causes, such as the varying atmospheric temperature and moisture. Furnace-men are obliged to carry a surplus of coke above the theoretical requirement, as a reserve source of heat to meet these frequently great and sudden changes in the atmosphere; and the existence of this reserve (which would not be required with dry-blast) prevents not only the close regulation of temperature, engine-revolutions, etc., but also the clear interpretation of their effects. It has been suggested, indeed, that a continuous automatic record of these factors would permit a better control of them; but such frequent changes (sometimes from hour to hour) as this suggestion would require in the engine-

Periods of Isabella Furnaces Nos. 1 *and* 3.

Temperature.				Ratio, Coke-Ore.	Tons of Ore Per Ton Pig.	Pounds Limestone Ton Pig.	Yield of Ore.			Charges Per Day.	Blast Pressure.	Flue-Dust Per Day, Tons.
Atmosphere		Dry Blast.										
Day.	Night	Day.	Night				Actual.	Theoretical.	Difference			
52.4	59.1	......	......	1 to 1.96	1.96	1,077	49.4	55.9	6 5	7 27	14.9	32.4
56.0	53.0	20.0	19.9	1 to 2.40	1.90	1,052	52.9	54.9	2.0	78.0	14.1	18.9
49.1	55 3	......	......	1 to 1.89	1.91	965	51.1	56.3	5.2	83.0	13.9	21.0
50 1	50.0	......	......	1 to 1.90	1.90	1,232	51.9	55.2	3.3	82.0	16.8	28.2
......	......	......	......	......	......	......	......	......	......	......	......	......
34.0	33.0	......	......	......	......	......	51.78	54.35	2.55	95.6	......	......
31.0	30.0	......	......	......	......	......	51.33	54.35	3.03	88.4	......	......
49.1	55.3	18.0	18.0	1 to 2.11	1.85	1,030	54.38	56.30	1 92	80.0 75.0	13.9	11.9

revolutions, blast-temperature, charge, etc., would be not only well nigh impracticable, but might also cause other evils, incident to irregular running, and greater than those which they were intended to remedy.

The natural atmospheric irregularities mentioned unquestionably affect the zone of fusion in the furnace; and such changes in the position and condition of this zone, which are the chief causes of difficulty in furnace-management, are, at times, by reason of atmospheric changes outside, much greater than has been generally assumed. Obviously, the wisest course, if practicable, is not passively to suffer these effects to begin and then try to remedy them, but to prevent them altogether by securing uniform blast-conditions, so that the zone of fusion may be held steadily in its position; thereby preventing scaffolding and irregular running, increasing the life of the lining, and, above all, permitting a reduction of the amount of coke otherwise required as a margin of safety. It is not necessary to amplify this argument. The conditions involved are familiar to all furnace-men.

The temperature of the atmosphere, as supplied to the engine-cylinders, is a very important factor in the use of dry air blast. The difference in weight of saturated and dry-air per cu. ft., at different temperatures, is shown in Table II.

TABLE II.—*Difference in Saturated and Dry Air Per Cubic Foot at Different Temperatures.*

Grains of Moisture at 100 Per Cent. Saturation.	At	Saturated.	Dry.
0.55	0° F.	equals 0.0863 lb. per cu. ft. of air,	or 0.0864
0.91	12° F.	equals 0.0841 lb. per cu. ft. of air,	or 0.0842
2.12	32° F.	equals 0.0805 lb. per cu. ft. of air,	or 0.0807
4.38	52° F.	equals 0.0772 lb. per cu. ft. of air,	or 0.0776
8.54	72° F.	equals 0.0739 lb. per cu. ft. of air,	or 0.0747
15.75	92° F.	equals 0.0707 lb. per cu. ft. of air,	or 0.0720

Table II. shows that a change of every 20° F. is equivalent to a change of 4 per cent. of the actual weight of the air pumped into the furnace per revolution, or every 5° F. is equivalent to 1 per cent. of the air by weight. Taking this on approximately 40,000 cu. ft. per minute blown at the Isabella works for one furnace, this 1 per cent. of difference in weight of air for every 5° would be equivalent to about 400 cu. ft. per minute, or a little more than the air blown in by one revolution. The effect of this can be shown in several ways. For instance, comparing Furnaces Nos. 1 and 3, the former running on dry-blast and the latter on natural air, both with the same air-cylinder capacity per min. (377 cu. ft. per rev.), we find that No. 1 ran 96 rev., or 36,192 cu. ft. per min., or 52,116,480 cu. ft. per 24 hr., dry-air blast entering at 28° (the weight of the air saturated at 2 grains of moisture being 0.081 lb. per cu. ft.), and gave 4,221,435 lb. of air in 24 hr., while No. 3 ran 113 rev., or 42,601 cu. ft. per min., or 61,345,440 cu. ft. per 24 hr., normal air entering at 90° (the weight of the air with 5 grains of moisture being 0.0711 lb. per cu. ft.) and gave 4,361,660 lb. of air in 24 hours.

In other words, about 3.75 per cent. less air goes in on 15 per cent. less blowing-requirement. As the temperature of the dry-air blast varies somewhat, and is usually lower, it can be accepted that practically the same amount of air enters by weight in the above stated conditions, thus materially reducing the power required.

To show more clearly the effect of uniformity of temperature, the data have been arranged in Tables III. and IV.

TABLE III.—*Some Results of the Dry-Blast Period of Furnace No. 1.*

1904.	Daily Pig-Iron Make. Tons.	Coke-Consumption. Pounds.	Moisture in Dry-Blast. Grains.	Moisture in Air. Grains.	Temp. of Dry-Blast. °F.	Temp. of Atmos. °F.
August,	448	1,753	1.80	5.62	27.1	71.0
September,	442	1,754	1.56	5.15	22.0	71.5
October,	416	1,862	1.18	3.11	18.0	55.5
November,	442	1,816	1.02	1.99	19.0	44.0
December,	445	1,823	1.03	1.43	17.5	33.5
1905.						
January,	428	1,822	0.76	1.46	13.5	30.5
August,	411	1,829	1.70	5.94	23.0	75.0

This indicates that, *per se*, the moisture-content is not so controlling a factor as the temperature, or rather the uniformity of temperature. The differences in temperature in dry air, though comparatively much less violent than those of moisture, are, of course, almost stationary compared to the differences in temperature of normal atmosphere as shown in the normal-blast period. As this matter appears of prime importance, I have arranged a similar statement of the dry-blast period of Furnace No. 3 in Table IV., in which the results appear of importance in showing that iron-production and coke-consumption are not entirely dependent on actual moisture-variations from month to month, in the dry-blast, or on even wider variations in the natural atmospheric air. They indicate, also, that the benefits secured are derived largely from the general uniformity in the moisture-content as well as the low degree of temperature of the air supplied to the blowing engines obtained through the use of dry-blast.

TABLE IV.—*Some Results of the Dry-Blast Period of Furnace No. 3.*

1905.	Daily Pig-Iron Make. Tons.	Coke-Consumption. Pounds.	Moisture in Dry-Blast. Grains.	Moisture in Air. Grains.	Temp. of Dry-Blast. °F.	Temp. of Atmos. °F.
February,	418	1,850	0.60	1.18	15.0	22.8
March,	407	1,837	0.95	2.26	14.0	43.8
April,	398	2,095	1.04	2.57	16.0	49.2
May,	436	2,024	1.50	4.08	23.0	62.2
June,	415	1,965	1.70	5.76	24.0	71.1

The actual lowering of both moisture and temperature is, of course, a very influential factor; but the variation of moisture within certain limits seems to be less important than the approximate uniformity in the weight of air going into the furnace,—the low temperature increasing the actual weight of air per revolution, or permitting the same amount of colder air by weight to be blown in by fewer revolutions.

Mr. R. A. Hadfield, in his recent Presidential address,[4] quoting an opinion that the economies ascribed to dry-blast "were conclusively shown to be due to increasing the burden of ore and the quantity of lime in the cinder, rather than to the use of dry-air," says:

> "It is well known that, by increasing the burden of ore on a furnace, the temperature of the escaping gases is lowered, the quantity of fuel required to smelt a ton of ore is reduced, and the yield of the furnace is increased."

Since all well-managed furnaces using natural air carry already as much burden as they can, a further economy by means of a simple increase of burden is out of the question, unless it be made practicable by the use of the dry-blast. Whether the direct cause of such further economy is the increased burden, or any one of the other conditions discussed above, it is only through the dry-blast that the result is made possible. Unless it can be shown that the Isabella furnaces, when operated with natural blast, did not carry as heavy a burden or as much lime as they ought to have carried under the circumstances, this criticism falls to the ground.

If any one effect of the dry-blast is to be emphasized as the chief cause of the resultant economy, it is, as already remarked, the uniformity of the conditions secured combined with as low temperature as possible (however they are characterized). To secure this all-important uniformity, it is evident that care must be taken to preserve unchanged the temperature of the air between the cold-air chamber and the engine-cylinder. This should be secured by properly covering the pipes.

The following comparative statements and tables are offered in additional support and explanation of the views above set forth:

In order to emphasize further the effect of uniform and steady dry-blast, both as to moisture and temperature, refer-

[4] *Journal of the Iron and Steel Institute,* vol. lxvii., No. 1, p. 58 (1905).

ence should be made to the record of Furnace No. 3 for January, February and March, 1905. During these months dry-blast was held very steady in moisture and in temperature, the former being between 0.5 grain and 1 grain, with an extreme variation of 0.5 grain, and the latter between 5° and 15° F. (average, 10° F.), with an extreme variation of 10° F.

The coke-consumption of Furnace No. 3 for these three months, with 50 per cent. Mesabi ore, producing basic low-silicon iron, was:—January (1905), 1,824; February, 1,815; and March, 1,787 lb.

After this period, the dry-air plant was not able to hold its moisture as low, or its temperature as steady, for the balance of dry-blast run on Furnace No. 3, the effect being shown in Tables V. and VI.

TABLE V.—*Moisture-Variation of Dry-Blast, Furnace No. 3.*

April,	1905,	0.5 grain to 1.75 grains	—Extreme variation :	1.25 grains.
May,	1905,	0.75 grain to 2.25 grains	—Extreme variation :	1.5 grains.
June,	1905,	1.25 grains to 2.50 grains	—Extreme variation :	1.25 grains.
July,	1905,	1.00 grain to 2.25 grains	—Extreme variation :	1.25 grains.

TABLE VI.—*Temperature-Variation of Dry-Blast, Furnace No. 3.*

*April,	1905,	10° to 25°	—Actual average 17°	—Extreme variation : 15°
May,	1905,	15° to 30°	—Actual average 25°	—Extreme variation : 15°
June,	1905,	20° to 30°	—Actual average 30°	—Extreme variation : 10°
July,	1905,	15° to 30°	—Actual average 23°	—Extreme variation : 15°

* Dry-blast conditions were better in April on account of the lower moisture of the outside atmosphere than in the following months.

The coke-consumption of Furnace No. 3 during these months, with 50 per cent. Mesabi ore, producing Bessemer high-silicon iron, was:—April (1905), 2,083; May, 2,032; June, 1,973; and July, 2,081 lb.

It will be noted that June, showing the lowest coke-consumption, has the least variation in the temperature of the dry-blast. The difference in fuel-consumption for these months between natural- and dry-air blast was less than the average practice, for the reasons given below.

The disturbing factors during these four months of 1905 were troubles at the furnace due to accidents and repairs, and to the leakage of bosh-plates. Moreover, the furnace used,

during that time, a large proportion of very fine ore, and made a higher silicon iron. During this period, the refrigerating apparatus was less effective, owing to the leaking of ammonia through the stuffing-glands. This was unsuspected for a long time, as the apparatus had worked perfectly for the seven preceding months, and many other supposed causes were first investigated. When the true cause was finally discovered, it was very quickly remedied, as is shown by the results in the succeeding months of Furnace No. 1 (the dry-blast being changed from Furnace No. 3 to No. 1 in August).

In practice, a variation of 0.12 per cent. in the silicon has been found to be brought about by a change of 1 per cent. of the burden. Producing basic iron with 1 per cent. and Bessemer iron with 1.5 per cent. of silicon, gives $0.50 \div 0.12 = 4.17$ per cent. decrease in the burden; or 2.08 per cent. increase in coke, since ore-burden to coke-consumption is approximately as 2 to 1. The average coke-consumption during the first three months of 1905 for Furnace No. 3 was about 1,800 lb. per ton of pig-iron produced; 2.08 per cent. of this is an increase of 37 pounds. In Furnace No. 3, the coke-consumption on Bessemer iron during April to June, 1905, was 2,080, or 280 lb. higher than in the first three months. Deducting from this 37 lb., due to the higher silicon, leaves 243 lb. more coke consumed, due to furnace irregularities and poor dry-blast conditions. As neither of the two factors of fine ore and high silicon changed the results materially when they occurred during natural-air periods, I do not consider them as very serious factors in raising the coke-consumption in this case; but the furnace conditions would affect the results seriously, as would also the poorer dry-blast conditions.

Comparing the foregoing with Furnace No. 1 while on dry-blast, we find the following:

TABLE VII.—*Moisture-Variation of Dry-Blast, Furnace No.* 1.

August,	1904,	1.50 grains to 2.00 grains—Extreme variation:	0.5 grain.
Sept.,	1904,	1 00 grain to 2.00 grains—Extreme variation:	1.5 grains.
Oct.,	1904,	1.00 grain to 1.5 grains—Extreme variation:	0.5 grain.
Nov.,	1904,	0.75 grain to 1.25 grains—Extreme variation:	0.5 grain.
Dec.,	1904,	0.75 grain to 1.00 grain —Extreme variation:	0.25 grain.

Very few variations above or below these figures.

TABLE VIII.—*Temperature-Variation of Dry-Blast, Furnace No.* 1.

Aug., 1904 (20 days),	15° to 30°—Actual average 22°—Extreme variation:	15°
Sept., 1904,	5° to 25°—Actual average 16°—Extreme variation:	20°
Oct., 1904,	10° to 20°—Actual average 18°—Extreme variation:	10°
Nov., 1904,	15° to 20°—Actual average 18°—Extreme variation:	5°
Dec., 1904,	10° to 20°—Actual average 18°—Extreme variation:	10°

Very few variations above or below these figures. In September, temperature and moisture variations were greatest, but temperature was very low and coke-consumption was 1,754 lb. for that month.

The coke-consumption on Furnace No. 1 with dry-blast on basic low-silicon iron, 50 per cent. Mesabi ore, during these months was:

In August,	1904 (20 days),	1,753 lb.
In Sept.,	1904,	1,754 lb.
In Oct.,	1904,	1,862 lb.
In Nov.,	1904,	1,816 lb.
In Dec.,	1904,	1,823 lb.

Comparing this with the higher coke-consumption on Furnace No. 3 from April to July, 1905, and the dry-air conditions prevailing during those periods, it would indicate that more uniformity and steadiness of dry-blast conditions existed during periods of lower coke-consumption, and that the low temperature and the low moisture-content of the dry-blast, combined with the least possible variation, appear to give the best results for low coke-consumption.

The difference due to temperature-variations under dry-blast conditions, as shown above, can be also shown as follows:—

At 18° F., 40,000 cu. ft. is 3,320 lb. of air per minute.
At 24° F., 40,000 cu. ft. is 3,276 lb. of air per minute.

That is to say, 44 lb. less air per minute (equivalent to 63,360 lb. per 24 hr.) are blown into the furnace by the same number of revolutions, under an increase of 6° F. in temperature. This represents about the difference between Furnaces No. 1 and No. 3 during the dry-blast periods.

This difference in the amount of air blown in would affect the running of the furnace as follows: Under given pressure and temperature of blast, the more air blown in per minute, the more rapid the combustion, and, consequently, the greater the heat of combustion, per unit of time. This rapidly increases the heat available for melting the material, and consequently diminishes the amount of actual fuel required to melt

the same amount of material, through greater intensity of heat for more rapid and complete combustion, and through a greater concentration of heat in the fusion-zone, bringing the latter closer to the tuyeres, where its best effect can be obtained. This is true, of course, always within certain limits, dependent upon the capacity and condition of each separate furnace.

The general conclusions reached through a study of all available trustworthy data may be summarized as follows:

1. Increased blast-temperature alone cannot account for a decrease of 20 per cent. in the coke-consumption, and an increase of 20 per cent. in the product of pig-iron. There was no significant increase in blast-temperature at the Isabella furnaces while they used dry-blast; and the records of those blast-furnaces which show a low fuel-consumption are not, in general, characterized by a blast-temperature sufficiently higher than that of the Isabella furnaces to account for their economy in fuel. This economy is doubtless due to innumerable local conditions,—such as character, richness and uniformity of ore, nature of fuel and flux, dimensions, shape and condition of furnace, perfection of machinery, intelligence and vigilance of management and working-force, etc., which every scientific furnace-manager takes into account in estimating the significance of his own experience, or comparing it with the experience of others. If all these elements of the problem could be made alike at all blast-furnaces, normal results of a given procedure could be at once determined, and any proposed change could be instantly weighed and judged. Such complete uniformity is obviously impracticable; but a device which operates in that direction, by removing all irregularity in a fundamental and most troublesome (though, hitherto, little regarded) factor, must recommend itself as an aid to those managers who have to deal with special local disadvantages in other respects, as well as to those who, by reason of fortunate conditions, are already making the best technical records.

2. The theoretically calculated effect of higher blast-temperature upon fuel-economy, etc., though possibly realized in isolated cases, is not confirmed in general practice, which indicates a considerably lower net result. Our modern blast-furnaces,

equipped with stoves for producing high temperatures of blast, have undoubtedly achieved much gain in product and fuel-economy, but not enough to equal the gains secured in those respects by drying the blast—gains, in fact, which can be added to those of such previous progress.

Thus, the proposition that an increase of 300° F. in blast-temperature would effect a decrease of 400 lb. in coke-consumption per ton of pig-iron produced is not confirmed by available records of practice, which indicate as the attainable saving from this source about 50 lb. of coke for each 100° F. of increase in temperature.

3. Of the furnaces using natural-air blast, it is usually the new ones that show high production with low coke-consumption. As the period of operation increases, the tendency to a reversal of these results becomes more and more evident. To this general rule, I have found few exceptions; and these were largely due to local conditions, not to be duplicated at other plants. This point seems worthy of consideration, because the results thus far reported as to Mr. Gayley's invention were obtained from furnaces already a long time in operation, and far from perfect in general condition.

4. The advantageous results obtained from the dry-blast are not sensibly affected by changes of season. This method is equally efficacious at all seasons of the year.

5. With due allowance for local conditions, it may be safely inferred that, at furnaces averaging more than 2,100 lb. of coke per ton of product, from 10 to 20 per cent. of gain in product and of saving in fuel can be effected by the use of dry-blast. At furnaces already using less than this proportion of coke, the gain may be somewhat smaller. To what extent, for instance, the fuel-consumption of a furnace already making, on natural-air blast, a ton of pig-iron with, say, 1,850 lb. of coke, could be further reduced by artificially drying the blast, is a question to be settled by actual experiment.

6. Apart from all other theoretical gains realized by the desiccation of the blast, my study of the records of practice impresses upon me the predominant importance of the uniform and regular running of the furnace thereby secured. This may be said to have been ignored in scientific discussions of blast-furnace practice before Mr. Gayley's paper appeared. The de-

sirability of removing moisture from the blast had been frequently suggested by technical authors, but the theoretical advantages to be thus realized did not seem to be commensurate with the expenditure involved. As Mr. Cochrane tersely put it, "the game was not worth the candle." It remained for Mr. Gayley to demonstrate a gain, not included in theoretical arguments, yet perhaps greater than any other therein considered: namely, that of uniform and perfectly controllable furnace-operation, and the consequent diminution of the amount of superfluous fuel hitherto carried as a "margin of safety," to meet the unforeseen contingencies of the "behavior" of the blast-furnace. This element has never been definitely measured in estimates of cost; but it is probable that every furnace manager would rate it above most other elements, as a factor of actual business cost.

Discussion.

H. M. Howe, LL.D., New York, N. Y.:[5]—The greater regularity of furnace-working, emphasized as the most important cause of the economies attending the Gayley dry-blast process, furnishes, in my opinion, an explanation which does not explain, since it leaves the question still to be answered, how and why the fuel-economy ascribed to this regularity can be so astonishingly great.

In the decades during which I have had to do with actual metallurgical manufacture as learner, then superintendent, then manager, then and still active director, the importance of regularity has indeed been impressed on me. It was long my habit to wake and time the revolutions of the engines against my pulse-beats, so as to assure myself of the needed regularity. But regularity, in and by itself, does not in the least explain why the fuel-economy which it gives is so astonishingly great. It simply pushes the difficulty one step farther back, leaving it really wholly unexplained. But the explanation is readily found if we take up the line of reasoning indicated by Mr. J. E. Johnson, Jr.,[6] and push it beyond the point at which he left it.

[5] Professor of Metallurgy, Columbia University in the City of New York.

[6] Notes on the Physical Action of the Blast-Furnace, *Trans.*, xxxvi., p. 454. Mr. Johnson called my attention to the importance of this line of thought several years before the disclosure, but not before the invention, of the Gayley process.

In brief, the economy of the dry-blast process, like that of the hot-blast process, is due essentially to its widening the margin between the temperature developed by combustion and what Mr. Johnson calls the "critical" temperature, *i.e.*, the temperature at and above which certain essential work of the blast-furnace process must be done. To the term "critical temperature" it may be objected that this term has already been appropriated by physicists for another meaning. The objection has some force, and a term without a dual meaning would be better. But after all "critical" is a broad, generic adjective, too valuable to be restricted to any narrow specific sense.

Adopting this term for lack of a better, it is true, as Mr. Johnson points out, that, for the work which has to be done above this critical temperature, the only heat available is that which the gases can give out in cooling to this temperature from their combustion-temperature, *i. e.*, the temperature to which the combustion in the hearth of the furnace raises them; because, once they have cooled to the critical temperature, their remaining heat is no longer available for work above that temperature. The economy of the dry-blast process, like that of the hot-blast process, is essentially due to the fact that each of these processes raises the temperature which the combustion in the hearth gives to the gaseous products of that combustion, and thus widens the margin between the combustion-temperature and the critical temperature; and it is the width of this margin that, in a rough way, measures the degree to which the heat generated can be utilized. If the critical temperature is 1,200° C. and the combustion-temperature 1,300°, there is a margin of only 100°; raise the combustion-temperature by 100°, or not quite 8 per cent., and you double the margin, and increase in a corresponding proportion the utilization of the heat.

It is like the case of water running over a waste-weir in the side of a canal. If the depth of the stream of water is originally 50 in., and the level of its upper surface 1 in. above the bottom of the waste-weir, and if the depth of the water is then increased so as to raise the level of its upper surface by 1 in., then, although the volume and depth of the stream as a whole are increased only 2 per cent., yet the water available for flowing through the waste-weir is increased 100 per cent.

The level of the waste-weir is a "critical" one; and the depth of water available for the waste-weir is the depth above that critical level.

Or, it is like utilizing the heat in a stream of hot gases to heat and boil water in a system of boilers and feed-water heaters, assumed for simplicity to offer such extended surface to the passing gases as to give ample opportunity for extracting from those gases all the heat which they are theoretically capable of giving up. If the water enters the heaters at 0° C. and is boiled at atmospheric pressure at 100°, then 100 calories are needed to heat each kilogram of water up to the boiling-point, and 536 calories to boil it. If now the hot gases enter the system at a temperature of 101°, they can give up to the work of boiling the water only 1°, or about 1 per cent. of their heat, because when they have cooled from 101° to 100° they have ceased to be able to transfer any more heat to water itself at the boiling-point, 100°. They then pass on to the feed-water heaters, still containing about 99 per cent. of their initial heat. But all the useful work that they can do in these heaters is to preheat enough water to take the place of that which they have just boiled away in the boiler. And as the heat needed in raising the water to its boiling-point is only 100-536ths, or about one-fifth, of that needed for boiling that same water, and hence as the work of preheating in the feed-water heaters is only about one-fifth that of boiling in the boilers, the heaters can use only about one-fifth as much of the heat of the gases as the boilers do, or only about one-fifth of 1 per cent. of the initial heat of the gases. So that the total percentage of the heat of the gases which can be utilized in the system of boilers plus heaters is only about 1.2 per cent. The rest of their heat the gases must carry away out of the system; and this heat must therefore be wasted.

If, on the other hand, the initial temperature of the gases is 636° C., then in passing through the tubes of the boilers they are capable of giving up to the work of boiling 536° of their heat, and of thus being cooled to 100°; passing thence to the feed-water heaters, they can there give up the remaining 100° of their heat to the work of preheating the water which is to replace that which they have just boiled away in the boilers. And, because the heat needed to preheat a kilogram of water is to that needed for boiling it as 100 is to 536, therefore the 100°

remaining in the gases as they enter the heaters is just sufficient to preheat to 100° the quantity of water which will exactly replace that which the gases boiled away in the boilers in their act of cooling from 636° to 100°.

Yet while Mr. Johnson's general explanation is correct, his ideas need to be much more carefully defined; and it is the need of this work of definition which leads me to prepare these remarks.

In order that the width of the margin between the working-temperature and the initial temperature of the gases shall be important, it is essential that the heat needed at and above that working-temperature shall be greater than at the average of lower temperatures. To make this clear, let us take the case of simply heating water up to its boiling-point, but without boiling it. Here, assuming that the specific heat of the water and of the gases respectively is constant and does not change with the temperature, then, so far as the theoretical efficiency of the heating-process is concerned, the heat of the gases can be utilized as fully if their initial temperature is 101° C. as if it is 1,001°. All that is necessary is that there shall be some margin between the initial heat of the gases and the temperature to which they are to heat the water; and the width of that margin is in one sense absolutely unimportant.[7] To heat 1 kg. of water from 1° to 100° takes 100 calories. If we take the specific heat of the gases as 0.25, their initial temperature as 101°, and their final temperature as they pass out of the heater as 1°, then each kilogram of gas, in cooling from 101° to 1°, gives out $100 \times 0.25 = 25$ calories, and therefore $100 \div 25 = 4$ kilograms of gas are needed for each kilogram of water. If the gas starts in at 1,001° and escapes at 1°, each kilogram of gas in cooling gives out $1,000 \times 0.25 = 250$ calories, and therefore only $100 \div 250 = 0.4$ kilogram of gas is needed for each kilogram of water; or one-tenth as much gas as in the previous case. But the heat in 1 kilogram of gas at

[7] For simplicity of presentation I have purposely ignored some very evident and important practical considerations, such as that the wider the temperature-margin between the gases and the body which they are heating, the more rapidly and thoroughly will the transfer of heat take place. The assumption that the walls of the boiler and of the feed-water heater conduct heat perfectly, and the tacit assumption that the outer walls of the furnace are adiabatic, of course, are made only to simplify the discussion.

100° is the same as that in 0.1 kilogram at 1,000°; and, so far as the needs of the operation are concerned, a given calorie can be used as efficiently in one case as in the other. This is essentially because the water, in each degree of its rise from 0° to 100°, needs the same quantity of heat; and the gases in cooling through each degree from 100° to 1° give out the same quantity of heat; so that, assuming that our walls are perfect conductors of heat, it is only a question of proper mechanical arrangements to transfer the heat given out by the gas in cooling from 101° to 100° to water which is simultaneously rising from 99° to 100°; and the heat which the gas gives out in cooling from 100° to 99° to water which is simultaneously rising from 98 to 99°, and so on. If this mechanical arrangement is perfected, the whole of the heat given out by the gases in thus cooling from 101° to 1° can be utilized and absorbed in heating water from 0° to 100°.

But, if the water has not only to be heated to 100°, but also boiled at 100°, then only the heat in the gases above 100° is available for that boiling; and, in order that the heat which those gases contain after cooling to 100° shall be utilizable in the work of heating the water up to 100°, the quantity of gas must be so small that the water in rising from 0° to 100° shall be capable of absorbing all that heat; and in order that the quantity of this gas which shall have boiled a given quantity of water shall be small, its initial temperature before it began transferring its heat to the work of boiling must be high.

Hence the need of a wide margin of temperature between the initial and the working-temperature in processes which, like the boiling of water, have a critical temperature; and the relative unimportance of the width of that margin in processes which, like the simple heating of water, have no critical temperature.

The reason why the importance of the margin between the initial gas temperature and critical temperature has generally been overlooked is that, in most of our heating processes, the margin is naturally so very wide that slight current variations in its width are unimportant. It is the natural narrowness of this margin in the blast-furnace process that gives to the hot-blast and the dried blast their very great value, and will some

day give a still greater value to processes for lessening the nitrogen in the blast.

From these considerations I deduce the formula given below.

If we let t_c = the critical temperature; t_i = the initial temperature of the gas (which is usually its combustion-temperature); h_c = the heat needed at and above the critical temperature, and h_b = the heat needed for raising the materials to be heated to the critical temperature; then in order that the heat shall be thoroughly utilized, and may be thoroughly recovered from the escaping products of combustion, it is necessary that the initial gas-temperature should be at least as high as is called for by the formula

(1) $(t_i - t_c) : t_c = h_c : h_b$, whence

(2) $t_i = t_c \left(1 + \frac{h_c}{h_b}\right)$.

Or, if we take into account the variations in specific heat, and let S_c = the mean specific heat of the gases between the critical temperature and their initial temperature, and S_b = their mean specific heat between the critical temperature and that of the atmosphere, then for formula (1) we should substitute:

(3) $(t_i \times S_c - t_b \times S_b)$; $t_b \times S_b = h_c : h_b$, and for equation (2),

(4) $t_i = \frac{t_c}{S_c} \left(1 + \frac{h_c}{h_b}\right)$.

The matter may be made clearer to some by looking at it in another way. Let us put ourselves in the position of one using the dry-blast process, and then see what effect on the fuel-consumption ought to be expected as the result of adding moisture to the blast. This not only calls for extra fuel to supply the extra heat needed for heating and dissociating this moisture, but (like cooling the blast or diluting it with additional nitrogen) by lowering the temperature which combustion develops, it narrows the already narrow margin between that temperature and the critical temperature, above which a very much larger quantity of heat is needed to make good the absorption due to the latent heat of fusion of iron and slag, and probably of deoxidizing silica and lime, than is needed at the average of lower temperatures. The combustion-temperature with moist-blast in Mr. Gayley's practice seems to be further lowered, and this margin thereby further narrowed, by a decided

lowering of the blast-temperature due to the fact that, with the hot-blast stoves used, the larger quantity of blast needed could not be heated so hot, although the gas used for heating it seems to have increased in quantity proportionally, and to have improved in quality. This part of the narrowing of the temperature-margin might perhaps be made good by enlarging the hot-blast stoves.

If the narrowing of this margin is to be atoned for by burning more fuel, then of the heat generated by that fuel only that corresponding to the narrowed margin between the combustion and the critical temperatures can be utilized. Of that represented by the range of temperature below this, we see no strong reason to expect that any important part can be recovered from the products of the combustion of this extra fuel by the descending solids, nor can the carbon monoxide generated by this extra fuel be converted into carbon dioxide by those solids, because, even with hot and dried blast, the power of those solids to absorb heat and to oxidize carbon monoxide appears to be already fully utilized, so that we have no strong reason to think that they can do more in either respect. This extra quantity of carbon monoxide and this extra heat may therefore be expected to be present in the gases when they escape from the top of the furnace; and these gases should therefore both be hotter and have a lower ratio of $CO_2 : CO$ with moist than with dried blast. Indeed, the quantity of carbon monoxide in the escaping gases should be even greater than this would imply, because, being hotter, their carbon dioxide should react the more energetically upon the entering coke, and thereby be reduced to carbon monoxide, and further increase the fuel consumption by thus dissolving away part of the entering coke, which must be replaced by more extra fuel.

Indeed, the matter is still worse. Narrowing the margin between the combustion and the critical temperatures lessens the thoroughness with which even the heat represented by the narrowed margin can be utilized even for the critical work, first because the cooler gases transfer their heat to the solid and molten matter less rapidly because of the narrowed temperature-margin between them, and next because, thanks to their greater weight and hence volume, they pass more rapidly by that matter. The heat-transfer is slower, and the time allowed for the transfer is shorter.

This statement in reverse, so to speak, is taken with some modification from a more extended discussion of the whole subject,[8] to which I refer those who may care for my views on this interesting question.

JOSEPH W. RICHARDS, Bethlehem, Pa.:[9]—Mr. Meissner's interesting communication confirms the statements of the former papers presented to the Institute by Mr. Gayley, and while practically proving the facts beyond peradventure, challenges the scientific world for a satisfactory explanation. The weight of opinion, as expressed by Mr. Gayley and others who have discussed the subject, is that the increased regularity of working of the furnace blown by dried air is the all-important cause of the economy obtained. This statement, however, is no explanation of the matter; and the question at once arises as to why increased regularity is obtained, and why irregularities occur if the blast be not dried. If a new administration of a railroad obtains better results by increasing the regularity of running of its trains, it is a true statement to say that the greater capacity of the railroad is due to increased regularity of operation; but the real question is, "How has the increased regularity been obtained?"

By regularity of operation is meant uniformity of temperature before the tuyeres, uniformity of grade of iron made, and uniformity of amount of output. It is admitted that the ordinary furnace is subject to irregularities in these respects; in fact, to such suddenly occurring irregularities that it is necessary to forestall them by providing always enough fuel in the charge to meet the worst irregularities which may occur. It takes about 24 hours for a change in the burden charged to have an effect at the region of the tuyeres, while serious irregularities may occur within 6 or even 3 hours, which it is therefore impossible to meet promptly by changing the burden. Under these conditions, it being necessary to be always "prepared for the worst," the blast-furnace manager is not foot-free to take full advantage of the most favorable conditions. If he can, therefore, be assured of uniformly favorable conditions,

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[8] *Iron, Steel and Other Alloys*, by Henry M. Howe, 2d edition, pp. 457 to 475 (1906). Cambridge, Mass.: Albert Sauveur.

[9] Professor of Metallurgy, Lehigh University, South Bethlehem, Pa.

such as drying the blast provides, he can dispense with the ordinary margin of insurance against unfavorable conditions, and run uniformly with the lowest possible fuel-charge. It is, therefore, absolutely true that the uniform conditions obtained by drying the blast do enable the blast-furnace manager to economize: the real solution or explanation of the ultimate "why?" has, however, still to be given.

The true logic of the situation is this: The uniform conditions obtained by drying the blast would produce no economies if they were not uniformly favorable conditions. If the conditions were such as to be uniformly the worst which could occur, the blast-furnace manager would have uniformity, but no one will maintain that he could economize thereby. Drying the blast produces uniformity of the favorable type; and this is the only kind of uniformity which will assist the manager. With blast not dried, the non-uniform conditions which cause unfavorable irregularities are the varying temperature and moisture-content of the outside air. With cold, dry air the furnace does its best; with warm, moist air it does its worst; sudden changes cannot be altogether foreseen; and, therefore, the manager must always be prepared for them by using a large fuel-ratio in the charge. The dry-blast provides the uniformly favorable condition, and therefore gives the basis for the economies realized. The whole question thus resolves itself into the query, "Why is the use of dried blast, of uniform composition and low temperature, better than the use of moist blast of non-uniform composition and varying temperature?"

The real explanation is two-fold. One reason is, that with blast of uniformly low temperature coming to the blowing cylinders, more oxygen is pumped into the furnace per day, thus increasing the rate of driving, other conditions remaining fixed; the second reason is, that the absence of moisture in the dried blast results in a higher temperature in the region of the tuyeres, thus increasing the reducing-power and smelting-down capacity, and incidentally inducing several other favorable conditions.

It needs no argument to show that, if the engines are running uniformly, the amount of oxygen drawn into the cylinders (independent of the amount of moisture in the air) is a function

of the temperature, varying inversely as the absolute temperature (from −273° C. or −458° F.) Air at 0° C., for instance, contains 10 per cent. more oxygen than air at 30° C., and therefore, with engines running uniformly, 10 per cent. more carbon would be burned at the tuyeres, per day, with air at 0° C. than with air at 30° C.—altogether aside from any question of moisture. If the air at the two temperatures named did contain identical quantities of moisture, the temperature at the tuyeres would be practically the same, in the two cases, and the ore smelted per day would be a little more than 10 per cent. greater with the cold air, because the greater speed of running would reduce radiation-losses and slightly increase the smelting-power of the furnace per unit of fuel burnt at the tuyeres.

The second reason, and the most important one, has been discussed at length by me in a paper already published.[10] I will here recapitulate the results of that analysis, referring for details to the original paper. Taking the two conditions, ordinary blast and dried blast, as described by Mr. Gayley, the temperature before the tuyeres is calculated as 1,861° C. with ordinary blast and 1,965° C. with the dried blast, a difference of 104° C. or 187° F., which difference results in an increased smelting-power and reducing-power in the region of the tuyeres. The absence of the moisture produces the higher temperature, there being available for smelting-down purposes 11,287 calories more, an amount equal to 23.4 per cent. of all the heat needed to smelt-down the pig-iron and slag. This extra heat, available just where it is most needed, while representing only 3 per cent. of the total energy of the coke put into the furnace, yet represents 23.4 per cent. of the energy of the coke which is generated and available for smelting-down purposes in the region of the tuyeres, for the simple reason that only from 20 to 30 per cent. of the power of the fuel is, in any case, available for smelting-down in the tuyere-region.

Another way of looking at the matter and arriving at a similar conclusion, is to itemize the saving obtained by using dried blast. The saving of fuel per 100 of pig-iron produced amounted to:

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[10] *Trans.*, xxxvi., 745–765 (1906).

	Per Cent.
On decomposition of moisture of heat,	2.95
On heat in waste gases,	5.25
On lessened radiation,	3.70
On better combustion of carbon to CO_2,	6.90
On smaller heat in slag,	0.10
On less heat in blast,	0.40
	19.30

This tabulation tells us just how and where the dry-blast effects its economies. The direct saving, by removing moisture, is relatively small, but this heat is saved at such a vital spot in the furnace, that the remaining 15 to 20 per cent. of economy follows as a necessary consequence. The amount of carbon burnt at the tuyeres, per ton of iron produced, being reduced by the higher temperature available, the amount of gases passing upwards per ton of charge is relatively decreased; therefore, their temperature on escaping is lower, and this lower temperature accounts for a saving of heat to the furnace amounting to 5.25 per cent. of the fuel ordinarily used. The smaller amount of carbon monoxide produced at the tuyeres per ton of iron reduced, produces better combustion above to CO_2 gas; for, since the ore has only a certain amount of oxygen to give up per ton of iron reduced, this oxygen will consume CO to CO_2, and the proportion of CO_2 to CO escaping must be larger, the smaller the amount of CO which there is in the furnace. This fact gives us the greatest single item of economy: 6.9 per cent. of saving in fuel being represented by the larger proportion of CO_2 to CO in the gases, because of the smaller amount of CO generated at the tuyeres. Radiation from a furnace is practically so much per day, and if we increase the output, the radiation per unit of product is decreased. This decreased radiation amounts to a relative saving of 3.71 per cent. of fuel.

Taking it all in all, therefore, we do not need to say that increased regularity explains the gains made by Mr. Gayley. The true answer is deeper seated: the dried blast produces regularity of the uniformly favorable type, as distinguished from an unfavorable uniformity. The reason why dried blast is more favorable than moist blast resides primarily in the fact of the higher temperature produced at the tuyeres. This smelts down iron and slag more quickly (increased rate of running) and more

economically per unit of iron and slag produced. This economy of carbon burnt at the tuyeres causes a better combustion or utilization of CO above the tuyeres, decreases the temperature of the issuing gases, and relatively decreases radiation-losses. These uniformly favorable conditions give an advantage over uniformly unfavorable conditions, and practically an equal advantage over irregular conditions varying from favorable to unfavorable, under which the manager must run, allowing all the margin necessary to meet the most unfavorable condition should it suddenly occur.

FRANK FIRMSTONE, Easton, Pa.:—It seems very clear that most of the fuel saved by drying the air is due to indirect advantages, and Prof. Richards's heat-balance shows very satisfactorily the several heads under which this economy is distributed and the value under each. I think furnace-managers will generally admit that, of several furnaces not differing very greatly in size and shape and using identical materials, the one which works with the greatest regularity almost always (always in my own experience) will show the lowest fuel-consumption. This follows chiefly from the fact that it is safe to burden such a furnace more heavily than can be done with one less reliable, a fact to which Mr. Gayley has himself referred.

It must be a great satisfaction to Mr. Gayley that his clear understanding of the probable gain, over and above the direct calculable gain from decreasing the water entering the furnace, has caused him to persevere to the attainment of such important results.

A word of caution may not be amiss in this connection. I believe it to be a fact that furnaces which tend to work regularly because of favorable conditions as to materials, shape or size, etc., show far less deviation from the normal, when subjected to a disturbing cause of given magnitude, than do those less favorably situated in these respects. I could cite fairly conclusive examples from my own observations. I have no personal experience with the materials now used in the Pittsburg district, but from conversations with blast-furnace managers from there, and from recent remarks in trade and technical journals, I think that furnaces which use them do not work with all the regularity desirable. If that be so, and admitting

what is said above as to the greater effect of disturbing influences under such circumstances, it may easily happen that materially less advantage will follow the application of dried air to furnaces more fortunately situated in these respects than those around Pittsburg.

It is worth noting that although according to Prof. Richards' heat-balance, the dry-air furnace receives 1,535 less calories from the hot-blast than does the moist-blast, a decidedly larger proportion of the total heat comes from this source than in the moist-air furnace (10.26 per cent. in the moist air, 11.21 in the dry air). This is of the greater importance, because heat in the blast corresponds to carbon in the coke burned to CO_2, and not, as does the heat produced in the furnace itself, in large part to CO.

We are apt, perhaps, to draw too positive conclusions from the apparent accuracy of the heat-balance, which after all is only a necessary consequence of the accepted doctrine of the conservation of energy. Not to speak of the present case, in which that portion of the gain arrived at indirectly greatly exceeds the part which could be pretty accurately calculated in advance, there have been plenty of instances in which a lucky change in methods of filling or more careful preparation of materials has resulted in gains almost comparable to those accomplished by Mr. Gayley, although what was actually done furnished, *a priori*, no grounds whatever to expect any change in the heat-balance.

R. W. RAYMOND, New York, N. Y.:—At the New York meeting of the Iron and Steel Institute,[11] I emphasized the uniformity of furnace-work secured by Mr. Gayley's invention as the chief element of its technical and commercial value; and I have seen no reason since that time to change this opinion.

Mr. Meissner has shown that the greater liability to wasteful irregularities involved in the use of a variable natural-air blast requires, as a necessary (yet still incomplete) safeguard, an excess of fuel, constituting, as it were, a reserve of heat for emergencies. This suggests a partial analogy, drawn from another branch of engineering, namely, the so-called "factor of safety,"

[11] October, 1904; see *Journal of the Iron and Steel Institute*, lxvi., No. 2, p. 301 (1904), and *Trans.*, xxxv., 1023 (1905).

employed in all constructions, by which the theoretically sufficient dimensions of a given piece of material are multiplied, in order to provide for unexpected strains or undetected weakness. In the building of the first Brooklyn Bridge, the factor of safety for the wires of the suspended cables was reduced considerably below the traditional figure, for the simple reason that the wire for those cables had been so carefully manufactured and so thoroughly tested, piece by piece, as to remove the danger of unsuspected weakness, and measurably to insure the behavior of the material in practice, according to the formulas of tensile strength and elasticity assumed for it. By such a legitimate reduction of the factor of safety, much money was saved in construction.

The extra coke, charged in a blast-furnace as a "factor of safety," is not, I confess, a perfectly parallel case. For the extra material put into wires or beams to provide for human oversight or exceptional emergency, simply stays there, drawing interest on its cost; whereas the extra fuel in a blast-furnace is burned, anyhow, whether it be needed or not. If it be not needed to meet sudden heat-requirements, its combustion must produce further irregularities in zones of reduction and fusion, grade of product, etc., or an increased proportion of CO and an increased temperature in the tunnel-head gases. This raises another question, into which I do not propose, at this time, to enter. We all remember that the illustrious Sir Lowthian Bell, whose memorable investigation and discussion of the blast-furnace process will be forever a classic authority, practically reached the conclusion (destructive at that time to many academic theories) that the blast-furnace process for the production of pig-iron was so admirably fitted for its purpose as to furnish, for a given ratio of CO to CO_2 in the escaping gas, the best internal smelting-results, together with sufficient and suitable fuel for heating the blast and raising steam in the boilers. Such a conclusion can never be final. The value of furnace-gases for use outside of the furnace itself (whether in making steam, in heating blast, or even—as Westman proposed—in calcining ore) has steadily declined in the opinion of iron-masters with the enormous increase in the product of each modern blast-furnace itself. In comparison with the quantity and quality of that product, they care much less now than formerly

whether outside hêat-requirements are supplied from the furnace-gases or not. There are many ways of getting heat for boilers, stoves and kilns, from fuel generally cheaper than that which is charged into the blast-furnace. And if any improvement in the smelting-process proper should, by reducing the amount of superfluous fuel used as a "factor of safety," require additional heat for outside purposes, we could easily help ourselves. Moreover, such an excess of fuel carried within the furnace is likely to do least good outside, just when it is most needed.

The immense practical importance of eliminating irregularities from metallurgical operations is illustrated in many other departments, besides those of iron and steel. To name only one example, the chief advantage of the modern reverberatory furnaces for the matte-smelting of copper-ores, the dimensions of which have paralyzed with astonishment those of us who painfully mastered the principles of copper-smelting a generation ago, seems to be the removal (through the presence of a vast reservoir of heat in the molten charge) of those frequent changes of heat-conditions, through charging and discharging, which were inevitable in operations on a smaller scale.

In the manufacture of pig-iron, the increased capacity of the blast-furnace has illustrated the same principle. A hundred little problems of the practice of the last generation have been, not solved, but actually drowned out of sight, by the high blast-temperature, rapid running, and great mass of stock in all stages of reduction, presented by our great modern furnaces. On the other hand, any irregularity of furnace-work means at least ten times as much loss of money as formerly. No matter how effective the scientific remedy may be, the manager of to-day cannot afford the cost of the temporary furnace-disease. His one prayer is, that he may know what he has to do, and what he is doing, and that he may not be overtaken by surprises—any one of which might cost him more than many theoretical improvements could recover for him.

Such calculations as Prof. Howe and Prof. Richards have given us are unquestionably of the highest value. But I do not think they discredit the importance of the principle of regularity as a source of economy. To use Prof. Howe's expression, they simply explain how and why regularity produces the results observed.

The argument of Prof. Richards suggests another analogy with constructive engineering, namely, that of the placing of material just where it is needed, instead of wasting it elsewhere—as illustrated by the old-fashioned wooden bridges built by rural carpenters, all parts of which were of the same size, or even by the famous Brooklyn Bridge itself, the four cables of which have the same diameter, though two of them carry more than twice as great a load as the other two. Such an economical distribution of material in a structure is practicable in proportion as the strength of each part, and the uniform amount or limits of variation of the strain upon it, are known. The necessity of a vaguely determined "factor of safety" is a hindrance. In a similar way, it seems to me that the measurable elimination of such a factor in the heat-supply of the blast-furnace permits the heat to be supplied at the point where it will be theoretically most effective, instead of at points where it may be accidentally needed, but is normally less useful, and sometimes, in certain respects, harmful.

But this is not all. Within certain limits, uniform conditions are, *per se*, more económical in practice than fluctuating conditions. The latter prevent a furnace from working up to any formula of efficiency that may be framed for it, no matter with what liberal allowances for contingencies. Nobody can tell what may be the loss of time, labor, material, quantity or quality of product, caused by a single episode of irregularity in working. Regularity does more than secure a regular income. It protects the furnace against sudden, premature death. Since the cost of re-lining, etc., for a new campaign must be repaid with interest by the product of that campaign, the length of the campaign is an important element of economy.

In short, I do not see that there is any essential difference between Prof. Howe and Prof. Richards, on the one hand, and those who have emphasized uniformity as the fundamental advantage of Mr. Gayley's invention, on the other. They analyze the advantages derived from a uniform dry-blast, and due to its dryness (*i.e.*, to the smaller amount of moisture, the larger proportion of oxygen, etc.); while we think there is a controlling practical advantage in its uniformity—a condition which their calculations necessarily assume. In other words, if we imagine a blast-furnace situated in a climate which furnished

a natural blast even lower, on the average, in moisture than Mr. Gayley's dry-blast, but varying, between day and night, from far below to far above that limit, we think the variations would more than counterbalance the practical economy of the more favorable average conditions.

At all events, I am warranted by this discussion, and by previous discussions in this country and abroad, in expressing the pleasure which, I do not doubt, is shared by all my American colleagues in the fact that the consideration of this new advance in metallurgical practice has already passed beyond the stage of skepticism as to its practical results, into the stage of serious, scientific inquiry concerning their explanation. This is the normal and proper course for such testing of new technical propositions. The exceptional feature in the present instance is the brevity of the initial period of incredulity, which I regard as a generous, and, at the same time, a just recognition of American representatives of both the science and the practice of metallurgy.

JAMES GAYLEY, New York, N. Y.:—Whether or not the increased uniformity obtained by the use of dry air explains the economies thereby effected, it makes possible, nevertheless, the various theories brought forward to account for those economies. It is a fact well known to all furnace-managers, that, owing to atmospheric conditions, there are a great many days and periods when the furnace could be operated much more economically than it is, the conditions being such as to permit of a material increase in burden; but instead of meeting these conditions, the blast-temperature is reduced, and often the volume of blast is increased, in order to gain a temporary advantage through increased output. A change in burden is not practicable, because, too frequently, before the larger burden could reach the tuyeres, a change in atmospheric conditions might occur and the fuel-consumption would be increased above the normal amount by reason of a derangement of the furnace. Thus, by reason of the lack of uniformity due to the ever-changing humidity of the atmosphere, advantageous conditions cannot be utilized, because they are temporary.

The drying of the air not only removes an unfavorable and varying element from the blast, but by keeping the content of

moisture in the dry air practically uniform, through refrigeration to a uniform temperature, it permits the delivery into the furnace of a practically constant weight of air, and therefore of oxygen—a result which cannot be obtained through an increase in blast-temperature. The elimination of the moisture removes a heat-extracting element at the tuyeres—a critical point in the furnace-operations—and this, together with the uniform weight of oxygen constantly supplied, intensifies the combustion, and localizes and lowers the fusion-zone. With natural air the fusion-zone is constantly fluctuating; when the zone is lowered, incipient scaffolds form on the area through which it is lowered; and when the zone is again raised, these scaffolds are melted loose, and, descending into the hearth, absorb heat. For this purpose alone, a margin of safety in fuel must be carried in the burden.

Before this process was applied practically, many calculations, made to determine the economic value of a removal of the moisture, indicated that, during the spring and autumn periods, a saving of from 3 to 3.5 per cent. in fuel could be obtained; while during midsummer a saving of 7 per cent. was possible. Yet the saving actually realized has been from 15 to 20 per cent., according to conditions. That uniformity, in the weight of air delivered to the furnace, is the greatest factor making for economy, is shown by the fact that quite as great a saving has been effected in the dryest months of winter as in the more humid months of summer.

Prof. Richards observes that the removal of moisture primarily produces a higher temperature at the tuyeres and thereby effects various economies. When Mr. Windsor Richards, an acute observer of metallurgical processes, visited the Isabella furnace, he immediately noticed and called attention to the intense brightness of the tuyeres. Prof. Richards hits the nail squarely on the head when he says that dry-blast "produces regularity of the uniformly favorable type."

Mr. Firmstone says there are many instances in which "a lucky change in methods of filling or more careful preparation of materials" has produced great and valuable gains in blast-furnace practice. I believe this to be true in his experience, which is to-day still greatly valued by all blast-furnace managers, and I know it to be true in my earlier experience; but I

do not think it is true of the practice of the past ten years. Mr. Firmstone remarks that, from conversation with blast-furnace managers in Pittsburg, with reference to materials there used, he thinks "that furnaces which use them do not work with all the regularity desirable." I think Mr. Firmstone has reference to the early use of Mesabi ores, with regard to which the statement is true. In a given unit of time, Mesabi ores will part with from 25 to 28 per cent. while the Old Range Lake Superior hematites will only part with 15 to 22 per cent. of their oxygen to form carbonic oxide gas; and this difference in reducibility has caused considerable irregularity in furnace-operations. (I question if anyone has so thoroughly investigated reducibility of American iron-ores on a practical scale as Mr. F. E. Bachman, of Port Henry, N. Y., from whom I have obtained my data.) But we have learned much in a few years; and under the practice of charging a furnace with ores of nearly uniform reducibility this trouble has disappeared. Moreover, it should not be overlooked that the economies effected by dry-blast, as reported in my previous papers, were realized with furnaces which had been using 50 per cent. of Mesabi ore for several years and working with as much regularity as I have found anywhere in my experience. The comparisons made by me were therefore fair in this respect, being based upon identical conditions—aside from the nature of the blast—in the records compared.

Let me suggest now certain economies, both practical and theoretical, in the application of dry-air blast. The plant at Isabella was designed solely to determine the value of dry-air blast, with a minimum of expenditure. Only one refrigerating-chamber was provided. It would have been better to divide this chamber by partitions, with an outlet from each division to the blast-main, that could be closed during the thawing-off period. Every day, one-third of the coils are thawed off; and, with the present construction, the quantity of moisture in the air supplied to the blowing-engines during this process is increased 50 per cent., as is shown in Fig. 1.

Let *A* to *D* represent 24 hr. We begin at *B* to thaw the coils and are running at 1 grain of moisture per cu. ft. of air; at *B* it increases to 1.50 grains. The thawing is completed at *C*, but the chilling-effects of that 0.5 grain of water (which repre-

sents the delivery into the furnace of 18 gal. of water per hr.) is not ended at *C*, but continues to a point represented by *E* or *E'*. We have therefore practically, under present conditions, out of the 24 hr. only 16 hr. of normal conditions as to moisture. By means of sub-chambers, we shall be able to run on a regular content of moisture, and I contend that the best work obtained at the Isabella plant, will, through the future employment of sub-chambers in construction, eventually prove to be the normal condition of work.

From theoretical calculations, it appeared that an economy in construction could be effected by conducting the operation in two stages; that is, by passing the air first over coils cooled to 30° F., and condensing a considerable portion of the moisture as water, after which, by passing it over a series of coils at a much lower temperature, it could be reduced to the desired temperature, while the moisture would be deposited as frost. Since the water from the first cooling would flow away, thus

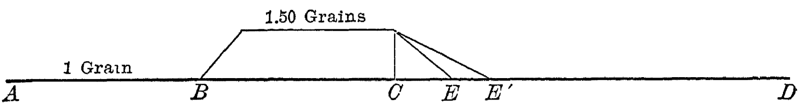


FIG. 1.—MOISTURE IN DRIED AIR, USING ONE REFRIGERATING-CHAMBER.

leaving a smaller amount of frost to be removed by thawing, this arrangement seemed advantageous. But when the data were supplied to manufacturers of refrigerating machinery, the estimates they furnished showed that there would be actually no economy in construction, and the saving in operation was indicated to be very small. The two-stage system appears to be more difficult of adjustment to varying conditions, and to involve more complications in operation—which are not desirable, since economy of power would depend on the proper adjustment of the two stages.

The use of ammonia by direct expansion in the refrigerating-chamber requires, in comparison with brine, more expensive pipes and fittings in construction, and more careful oversight in adjusting the valves to secure a uniform refrigeration. According to theoretical calculation, the direct expansion shows an economy over the brine system; but I doubt whether this would prove true in practice. Our first installation with direct expansion at the Isabella furnace showed that additional

labor would be required. We are now using, with the brine-system, two men on day-turn and one man at night, while with direct expansion four men would be required for the two turns. Direct expansion presents, moreover, additional opportunity for increased leakage of ammonia.

My first experiment was conducted on the two-stage system. The air was taken in a heated engine-room in winter, and cold river-water was passed through one chamber and ammonia directly expanded through another; and (in the last experiment before the installation at the Isabella furnace) two large chambers were used, in one of which brine was circulated, while in the other ammonia was directly expanded.

It has been suggested that the cold air passing from the refrigerating-chamber could be conducted to a regenerator, to cool the incoming air. This would increase the cost of construction, and does not hold out much promise of economy in the refrigerating operation. The dry air, being heated by this operation, would require more revolutions of the blowing-engines to deliver the required weight of air to the furnace; and this would consume more power. But the chief objection is, that the dry air, absorbing heat from the incoming air, would vary in subsequent temperature nearly as much as the natural air, and the process would thus, to a considerable extent, revert to natural-air conditions. Any process for extracting moisture that does not also provide for the delivery of a practically constant weight of air to the furnace, will not yield the economy that is now obtained at the Isabella furnace.

Our experience with the use of dry air shows that, even under adverse conditions, great economies are possible. Furnace No. 1 at the Isabella plant has made a large output, and has been banked three times—a process which is always detrimental to economy. At the present time, a large part of the lining has given way at about the middle of the stack for a space 20 ft. high and reaching half way around the jacket. A constant stream of water is required to play on this portion of the jacket, in order to keep the furnace in operation. Yet, having all this to contend with, No. 1 is producing iron under dry-blast with a coke-consumption 340 lb. smaller per ton of pig than its companion-furnace, of the same size and supplied with the same ores and fuel, under natural-air blast.

The experience of 18 months with dry-air blast has demonstrated several things:

1. The charges settle more quickly and uniformly after casting than in a furnace blown with natural air.

2. The loss in flue-dust is about 2 per cent. less.

3. The furnace works with greater regularity, and off-grades of iron, which are usually sold at a considerable reduction in price, are seldom made; for dry-air blast maintains the regularity of the grade of output, which, in a merchant-furnace, is a considerable advantage. In the product of a furnace blown with natural air, when the silicon is lowered, the sulphur often is increased to such an extent that the pig must either be remelted, or sold at a material reduction in price. With dry-air blast, we have found that there is rarely an appreciable increase in sulphur corresponding to a material rèduction in silicon.

The ability to produce pig-iron according to standard specifications, with increased uniformity of grade and composition, makes this process specially valuable to merchant-furnaces, in the operations of which a considerable profit is habitually sacrificed in the marketing of "off" grades of pig-iron.

From the standpoint of a practical blast-furnace manager, retaining a vivid recollection of the daily whims of that crude yet delicate apparatus, the blast-furnace, I can heartily indorse the views expressed by Dr. Raymond, while accepting with respect and gratitude the theoretical explanations advanced by Prof. Howe, Prof. Richards, and other eminent authorities at home and abroad.

Piping in Steel Ingots.

BY N. LILIENBERG, PHILADELPHIA, PA.

(Bethlehem Meeting, February, 1906.)

During the past few years, the requirements for steel have been raised so high that soundness is more important than ever before. The old practice was to make steel ingots of sufficiently large sections to permit a considerable reduction of area in rolling and hammering to a given size, relying on the fact that by such rolling and hammering all cavities will be squeezed together and practically eliminated. The finished sizes, if found by inspection of the surface to be faultless, were therefore pronounced to be sound. But when such steel is "worked up," the manufacturer is surprised and disappointed to find that it splits, or that streaks and seams make the product unmarketable. Large sections, such as rails, beams or heavy forgings, may, within certain limits, be sufficiently strong, even if the sides of the cavities are only squeezed, and not welded, together. But it is well known that, in the interior of heavy articles, under certain conditions, forging may develop small streaks into large cavities. Small sections, steel for ordnance and tubes, and numberless articles in the tool-trade, always show up the defects. It may be doubted whether cavities in steel ingots can ever be completely welded. The old theory is, that the silver-clear blow-holes, without the blue oxide, can be welded by sufficient working. They are, however, not empty, but are filled with carbonic oxide. It is admitted that the "blue" cavities, resulting from air mechanically drawn down into the mold during the casting, can never be welded, because the oxide does not melt, and there would be no escape for it if it did. The only way in which steel can be made satisfactory for the above-named particular purposes is by having it "dead molten." But then the trouble of piping is encountered.

As is well known, a pipe is the result of the contraction of the liquid interior after the surface of the ingot has become solid by contact with the mold. When this contraction is partly

taken up by blow-holes, there is, of course, less sinking of the metal at the top. The soundest steel, therefore, has the worst pipe. Some manufacturers having found that, for certain purposes, piping is a worse evil than blow-holes, are, therefore, purposely casting honey-combed ingots. Several manufacturers rely on casting steel at such a temperature that the blow-holes arrange themselves in a zone midway between the center and the surface, thus making the defects invisible after rolling or hammering. But, aside from the impossibility of always arranging the gas-bubbles in this way, the defects will afterwards appear in working, as above mentioned. The pipe is always coated with blue oxide from contact with the air. This oxide is formed instantaneously; and therefore the common practice of filling up the pipe with liquid steel during casting and solidification only forms a stopper which is not welded to the ingot.

I. The Prevention of Piping.

A great many processes have been invented to obviate piping. Those of importance may be arranged in two classes: keeping the steel liquid, and compression of the steel.

1. *Keeping the Steel Liquid as Far Out Towards the Mold as Possible, in Order to Get an Even Sinking of the Top Surface.*

All methods working on this principle suffer from the evil that, the period for solidification being prolonged, the separation of the impurer, and therefore lighter, steel has more time to take place, and the segregation will be more pronounced. Under this heading the following methods may be mentioned:—

(*a*) *Addition of Aluminum*, either in the ladle or in the molds. The aluminum increases the heat by combining with part of the dissolved oxygen, and, at the same time, lowers the melting-point of the steel (which, of course, works in the same direction). It is well known, however, to-day, that aluminum does not realize the expectations formerly entertained concerning its use. It makes somewhat sounder steel, but it does not prevent piping; and it has been found, in many instances, to have the tendency to throw a bridge of solid, sound steel over the top of the ingot, thus hiding a large unseen cavity in the interior.

(*b*) *Addition of Thermite.*—This mixture of aluminum and iron oxide gives a greater heat than aluminum alone, on account of the extra supply of oxygen present; and its use makes the steel in the mold fairly sound. But, inasmuch as the higher temperature cannot prevent the steel from solidifying first in the outer layers, a pipe will nevertheless form, although it will be somewhat smaller.

(*c*) *Use of Clay-Lined Funnels.*—These, placed on the top of the molds, shorten the pipe, because the steel does not chill so rapidly against the less-conductive clay as against the bare mold. The upper part has then to be carefully cut away so far down as to leave no part of the pipe. The exact distance from the top, at which the ingot should be cut, is somewhat difficult to determine in this case as well as in the common practice of cutting-off the upper part of the ingot and leaving only the lower part for use. In both cases the production is burdened by the considerable cost of fuel, labor and waste involved in remelting the upper or "piped" portions of the ingots, or of selling these portions at the scrap price, which is below cost.

(*d*) *Casting by Overflow from One Mold to Another.*—In this method the molds are arranged in a row close to one another; and while tolerably sound ingots are obtained, there is the disadvantage that much scrap is formed by the steel running through the conduits. It is also obvious that the steel coming to the last mold is much cooler than that in the first.

(*e*) *Use of Electricity.*—This method has sometimes been used to heat the upper part of the mold, and thereby prevent the chilling of the steel in contact with the sides before the center gets solid. It is obvious, however, that, in order to be effective, the heat would have to be so great that the mold would be destroyed by the steel, which would readily attack the walls. Moreover, the method is considerably more costly than that of cutting off the piped part.

2. *Compression of the Steel in the Molds.*

Much has been said and written about liquid-compressed steel, and the general opinion seems to be that, aside from the obliteration of the pipe, little or nothing is gained by compressing liquid steel, which is absolutely sound and "dead" molten. Since liquids in general are inelastic, and therefore incompres-

sible, the possible degree of compression for liquid steel depends on the included gases. If the press is arranged so that these gases, which are elastic, have no avenue of escape, the effect of the compression will only be to leave cavities of smaller size. I need hardly say that the compression of liquid steel in the mold has quite a different effect from that of forging the ingot afterwards. In the latter case, the object is to change the structure of the steel at the same time with its shape. Among the numerous devices for compressing steel in molds the following three methods may be mentioned.

(*a*) *Compression from the Top*.—While there is no difficulty in this method, if the whole charge goes into one mold, the machinery will be considerably complicated if several ingots of smaller sizes have to be cast. These ingots will have to stand on trucks and move in procession under the press; or several presses will have to be arranged, with additional complications It is almost impossible to time the casting and the pressing in. harmony. If the latter is done too early, the cavities will continue to form in the interior of the steel. If too late, the pipe is already formed and oxidized, and therefore will only be squeezed together, but not eliminated. Any compression from the end of an ingot is attended with the danger of tearing the interior loose from the surface. This process is in use in some places in England and Germany.

(*b*) *Compression from the Bottom*.—This idea originated in practice at St. Etienne, France; and the process is in use at several French works. It naturally suffers from all the disadvantages of the top-compression, and, in addition, the hydraulic machinery is contained in a deep pit, which makes it difficult to be cared for, and also exposes it to slag, dirt, and accidental spurts or leaks of liquid steel. In both cases the molds have to be made sufficiently strong, and correspondingly heavier and more expensive. It is, of course, a special and simpler case where the whole charge is poured into one mold, as, for instance, in making a gun of large size.

(*c*) *Compression from the Sides*.—In this method the top is left open, permitting the gases to escape, and the pressure is applied directly in proportion to the formation of the pipe, so as to keep the ingot always full to the top. It is obvious that the pressure can be better timed in this way than in the preceding cases.

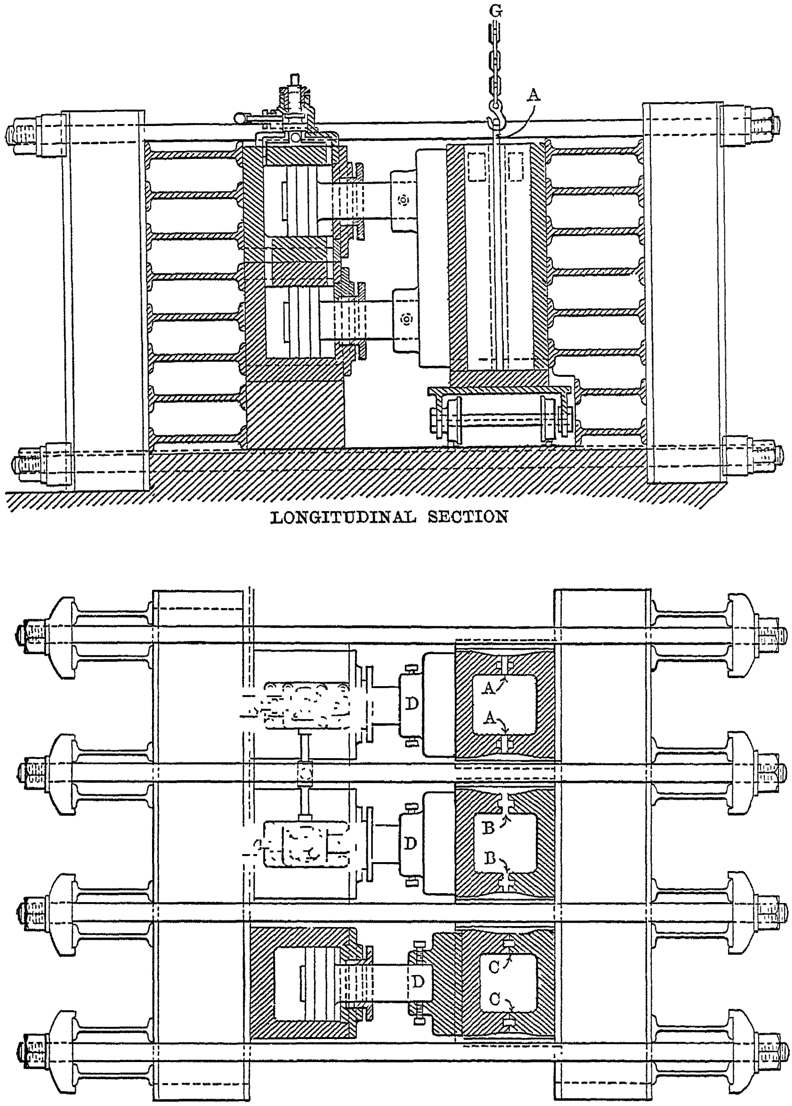
Some years ago an apparatus of this kind was built in Pittsburg, having four slowly-moving rollers in the same plane. The ingot, with liquid interior, was taken from the mold and placed between the rollers; and as it slowly moved downward the compression kept the liquid steel toward the top, and prevented the formation of a pipe. This is undoubtedly the right principle; but it encounters the insurmountable practical difficulty of transferring the semi-liquid ingots from the mold to the rollers, where they are also liable to be warped and twisted. Moreover, if liquid steel be shaken, it is liable to become honeycombed by the escapeof part of the gases dissolved in it.

Side-compression in a stationary mold, having none of the objections above mentioned, is now successfully carried out, even with the largest-sized ingots. This has been mainly accomplished by Mr. John Illingworth, who, for a long period, has devoted his large experience to the practical solution of the problem, and has obtained many patents in this field.

Mr. Illingworth began by casting a continuous ingot through a prolonged vertical mold, the lower half of which was cooled by circulating water. The ingot dropped down through the bottomless mold and was broken off by a hydraulic clutch while still red-hot. This method, which worked very well for small ingots, presents difficulties for the larger sizes. The second group of Mr. Illingworth's inventions comprises casting-machines, built on the principle of hydraulic rams, which squeeze the two halves of the mold slowly together after they have been sufficiently opened to enter a plate between the ingot and the mold. This plate, having the section of a segment of a circle, displaces a corresponding volume of the semi-liquid interior. The two halves of the mold are drawn apart as soon as a thin skin is formed, holding the liquid steel during compression. The movement of the molds, which are placed close together, is made by draw-bars connected with the head of the hydraulic rams. This apparatus has given very satisfactory results for small ingots cast from crucibles, but it is not well-adapted for casting from a bottom-tapped ladle into large molds. A special apparatus on the same principle was built for this purpose, but the experiments were not continued sufficiently long to secure successful results.

The problem of compressing large-sized ingots, cast from a

ladle, is, however, now completely solved. Mr. Illingworth, together with Mr. S. Robinson, has built such casting-machines



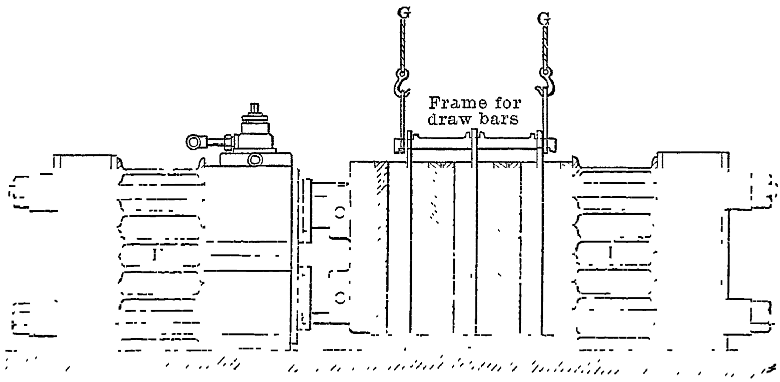
PLAN

FIG. 1.—ILLINGWORTH CASTING-MACHINE FOR LARGE INGOTS IN MOLDS ON WHEELS.

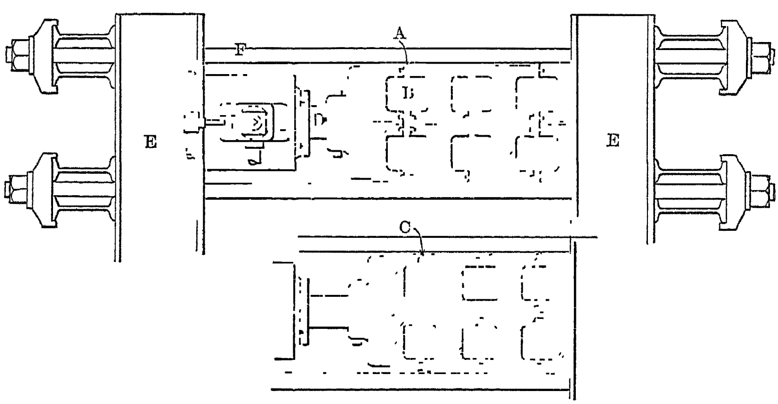
at the steel-works of Jessop & Sons, Sheffield, England, which uniformly turns out ingots of the largest size, absolutely solid from top to bottom.

The half-tone photograph of split ingots, Fig. 4, showing one with pipe, *A*, and another which has been compressed solid from end to end, *B*, is both interesting and convincing.

There are two kinds of casting-machines, one for large and the other for small ingots. Figs. 1, 2 and 3 illustrate the construction and working of these machines, which are covered by U. S. Patent No. 810,654, issued Jan. 23, 1906. The principle is



LONGITUDINAL SECTION.



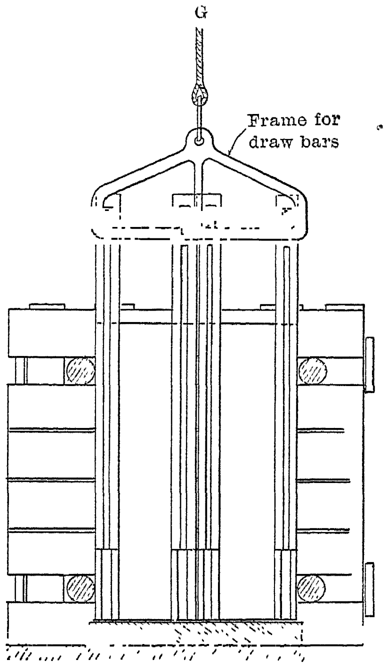
PLAN.

FIG. 2.—ILLINGWORTH CASTING-MACHINE FOR SMALL INGOTS IN GROUPS.

to cast the steel in molds divided in halves and held together, during the casting, by hydraulic pressure. In the planed side-edges of the molds are grooves which admit bars of a cross-section shown at *A*, Figs. 1 and 2. After the metal has been poured into the mold, sufficient time is allowed for a crust to form, while the interior is still liquid. The bars are then withdrawn by chains, *G*, suspended from an over-head hoist, which

leaves an empty space between the two halves of the mold, as shown at *B*. Hydraulic pressure is then applied from the ram, *D*, and the outer half of the mold is moved forward slowly, so that the liquid interior is always kept full to the top until the mold is closed, as shown at *C*. The volume of the bars corresponds as near as possible to the volume of the cavities which would otherwise be formed.

The large machine, shown in Fig. 1, is for large ingots cast



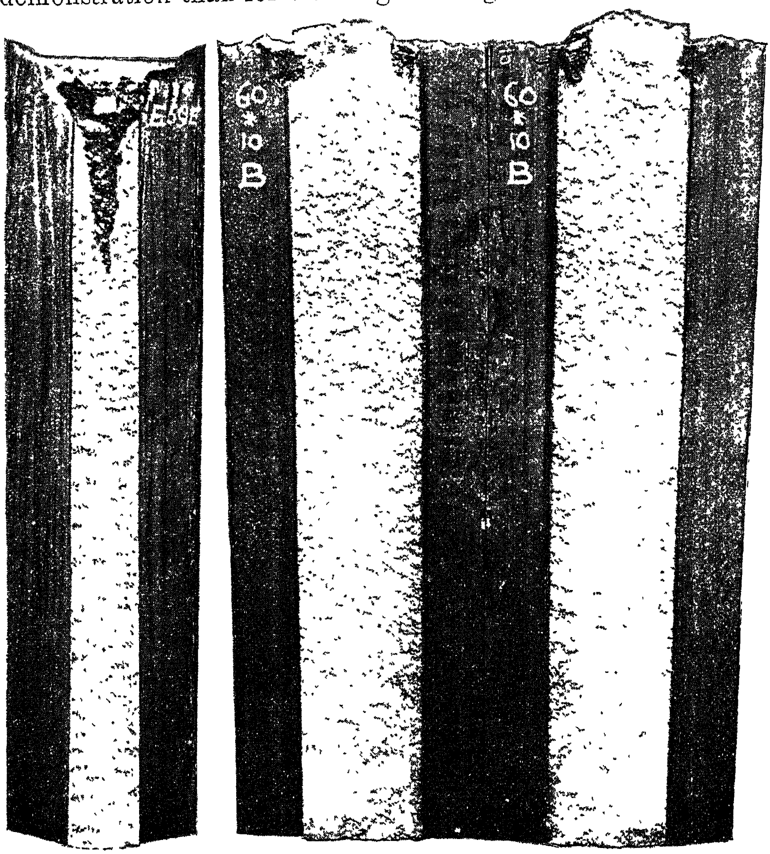
TRANSVERSE SECTION.
(Scale double that of Fig. 2)

FIG. 3.—ILLINGWORTH CASTING-MACHINE FOR SMALL INGOTS IN GROUPS.

on trucks, which are transported out of the foundry immediately after casting, each ingot to be compressed separately. This action can be so modified that three 2-ton molds can be compressed at one time.

The small machine, illustrated by Figs. 2 and 3, is designed to compress ingots of smaller sizes that are cast in groups on the ground. In this case the casting must be made simultaneously in all the molds in order to have the same initial temperature. This can be effected either by bottom-casting or

by pouring the steel from the ladle through an under hanging-trough provided with a hole for each mold. With bottom-cast ingots the gates break off during the movement of compression. These illustrations have been prepared more for demonstration than for working-drawings. For clearness, sev-



A
Cast and cooled in ordinary solid mold. Shows piping at upper end.

B
Cast and compressed in sectional mold during cooling. Shows absence of piping.

Fig. 4.—Steel Ingots Split Longitudinally to Show the Elimination of Piping by Compression During Cooling.

eral details of construction have been omitted. These details have to be modified to suit each particular case.

The hydraulic cylinders on the one side of the press and the back of the molds on the other are supported by six I-beams (*E*, *E*, in Fig. 2) laid horizontally on pieces formed to fit them, and held together by vertical beams and screw-bolts.

In regard to the details of working the Illingworth press, 3 or 4 men can cast and compress 60 ingots, 13 in. square and 40 in. long, in an hour's time. The edges of the molds have to be planed, but a mold can be used for about 125 heats before this repair is needed.

It is impossible to give general statements of the cost of operating this press, since the economy would have to be calculated for each particular case.

By careful investigation it has been shown that:—

1. During compression the surface of the ingot is neither crushed nor folded, but is merged in the mass, the same as in the roughing-mill, although the circumstances attending compression are not so favorable as those of rolling.

2. No groove is formed between the joints of the mold. If, with special steels, there should be some tendency to this, it is easy to make the edges of the bars slightly convex, thus leaving concave grooves in the ingots, which, by compression, straighten out to a plane.

The Beard-Mackie Sight-Indicator for the Measurement of Marsh-Gas in Collieries.

BY M. H. HARRINGTON, PHILADELPHIA, PA.

(Bethlehem Meeting, February, 1906.)

THE *Transactions* of the Institute afford abundant evidence of the general recognition by mining engineers of the importance of a safety-lamp which will not only give warning of the presence of fire-damp, but will also indicate the proportion in which it is present.[1]

Mining-men are only too familiar with the frequent and sudden changes that habitually occur at the working-face of a gaseous mine. Nothing illustrates more forcibly the pregnant condition of the strata enfolding the seam and the seam itself, than the fact that a circulation of perhaps 150,000 cu. ft. of air

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[1] See *Trans.*, xiii., 129 (1884–5); xiv., 410 (1885–6); and especially the elaborate papers of Profs. Chesneau and Clowes, xxii. (1893), 120, 606, with the discussion, xxii, 725 (1893).

may contain 2 or even 3 per cent. of marsh-gas, representing a volume of from 3,000 to 4,500 cu. ft. of this gas pouring out each minute from the natural strata into the mine-workings. It would, indeed, be strange if this great outflow of gas were always constant, and it creates no surprise that such earth-breathings, as they may be called, are spasmodic and irregular in their wane and flow. There are other causes that act to vary the gaseous condition of mine-air. Gas has a tendency, often very manifest in mine-workings and airways, to stratify and, at times, to travel long distances in veins or streams. It is common to catch and lose the gas in the lamp when making a test. At times two lamps, held side by side in an air current, will indicate different gaseous conditions of the passing air. When testing for gas, the lamp may suddenly fill with flame, in a mine-chamber, where a moment before a careful test showed the presence of but 1 per cent. of gas.

It is sometimes remarked that there is little need of detecting a smaller percentage of gas than is revealed by the common Davy lamp. This quantity, according to the ability of the observer to discern the flame-cap, will vary from 2.5 to 3 per cent. Indeed, few men are able to discover the slightest indication of gas when the quantity present is as low as 2.5 per cent. In a recent discussion of this subject that took place in a meeting of the Institution of Mining Engineers, England,[2] it was stated that the observance of a minute flame-cap by two individuals will differ according to the object each had in view. An instance was cited where a mine-inspector and a mine-overman, both experienced mining-men, entered a place where the inspector wanted to show the presence of gas; the overman failed to detect the flame-cap, which was plainly visible to the inspector. Shortly after, the inspector failed to see any appearance of gas on the flame in the overman's lamp in another heading where the latter claimed he had found gas. As a matter of fact, there is too much guessing in the method of detecting the presence of gas by the height of the flame-cap, which is very indistinct, except when the proportion of gas reaches 3 per cent.

The papers of Profs. Chesneau and Clowes describe many

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[2] *Transactions of the Institution of Mining Engineers*, vol. xxvi., p. 217 (1903–4).

ingenious devices; several of these, while possessing more or less accuracy, require, as a rule, peculiar and sometimes expensive apparatus. It is the purpose of my paper, not to discuss the relative merits of these devices, but to call attention to a very simple attachment, known as the Beard-Mackie sight-indicator, which was designed three years ago, after a long series of experiments, by Prof. J. T. Beard, principal of the coal-mining department of the International Correspondence Schools, Scranton, Pa., but formerly an active colliery-manager and engineer in the Western coal-field, and secretary of the Iowa State Mining Board. This little apparatus, which can be adapted and attached to any ordinary safety-lamp, is intended to utilize the familiar phenomenon of the lengthening of the flame-cap in proportion to the increase of methane in the surrounding atmosphere in such a manner as to measure the height of the cap with exactness, and to eliminate the element of guessing above mentioned, by substituting a plainly visible indication of the position of the flame-tip, whereby the percentage of gas in the air is made known.

This indicator has been successfully tested in practice both in this country and abroad. The following description of the indicator was written at my request by Prof. Beard:

"The device depends upon the well-known property of platinum of inducing the union of oxygen and other gases in contact with its surface, which property is possessed by the compact metal to a less degree than spongy platinum only in proportion as its surface is less than the surface of the latter. The stimulation of the chemical activity at the surface of the metal, when gas is present, is sufficient to maintain a red heat in a platinum foil. This is shown by heating the foil to redness in a gas-flame, and then shutting off the gas suddenly and at once turning it on again. While the heat developed, in this case, is not sufficient to cause the gas to ignite, it maintains a red glow in the platinum, which only ceases when the gas is shut off. The degree of heating is determined by the ratio of the surface to the volume of the metal, which, in the case of spongy platinum, is sufficient to cause the ignition of the gas, and which, in the fine wires of the indicator, causes a white heat that enables the lamp to hold its flame, relighting the

lamp after extinction in 'sharp' gas. This feature, which is of great advantage in the work of testing for gas in the mine, has often been proved, both in the mine and in the laboratory, when the lamp has been introduced into an atmosphere of gas that was extinctive owing to the excess of gas. All flame would then die out within the lamp, the wires alone remaining incandescent, and relighting the gas when the lamp is slowly withdrawn, the gas in turn relighting the lamp. It is noteworthy that the platinum wires incandesce at a considerable height above the tip of the flame, owing to the peculiar property of the platinum that stimulates the combustion of the gas in contact with its surface.

"The underlying principle of this device is practically the same as that involved in the Liveing indicator, described by Prof. Chesneau;[3] but in the Liveing indicator the original source of heat is an electric current made to pass through the platinum wire, while in the present device the heat is derived originally from the flame of the safety-lamp in which the indicator is placed.

"The same principle had been applied in a less practical manner, previously, by Mr. M. D. Mackie, an experienced fire-boss at the Marvine colliery of the Delaware and Hudson Railroad Company's coal department. Mr. Mackie supported a small platinum coil above the flame of a common Davy lamp by means of a brass rod that passed up through the oil-vessel into the lamp. By moving the rod up and down, a position was found in which the platinum wire just ceased to glow. The height of the wire above the flame at the moment it ceased to glow furnished the means of telling the percentage of gas in the air fed into the lamp.

"After seeing Mr. Mackie's lamp, and making a few preliminary experiments with wires of different gauge, I designed the present form of the sight-indicator; but not wishing to deprive Mr. Mackie of the credit that is his due, I have called it the Beard-Mackie sight-indicator.

"The device, as shown in Fig. 1, detached from the lamp, consists of a ∩-shaped support, made of No. 14 brass wire and riveted to a brass disk, which forms its base and fits over the

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[3] *Trans.*, xxii., p. 139 (1893).

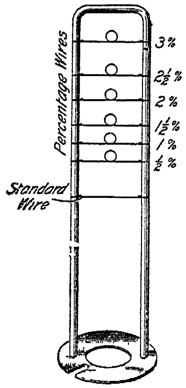

FIG. 1. — Wire-Frame Support for Platinum Wires.

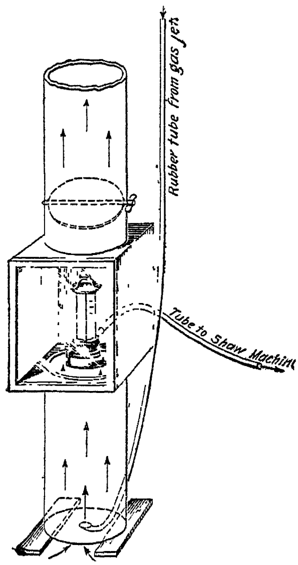

FIG. 2.—Box and Davy Lamp Used in Calibration.

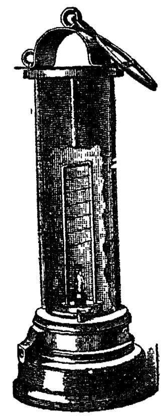

FIG. 3.—Sight-Indicator in a Davy Lamp.

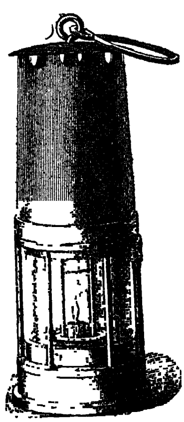

FIG. 4. — Sight-Indicator in a Bonneted Clanny Lamp.

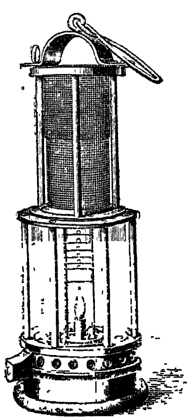

FIG. 5.—Beard's Special Form of the Clanny Lamp.

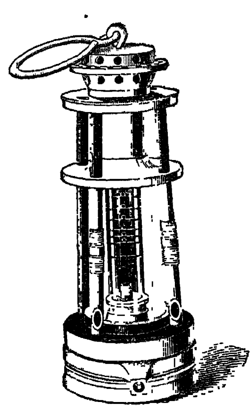

FIG. 6.—Special Form of the English Lamp Adapted for the Sight-Indicator.

wick-tube of the lamp. The indicator is thus held firmly in position in the lamp by the same nipple that secures the burner. As shown in Fig. 1, supported on this frame are seven platinum cross-wires, a straight standard-wire at the bottom, and above this, six percentage-wires, each having a small loop or circle in the middle. The loops enable the incandescent-wires to be more readily and quickly discerned within the gauze of the lamp. The lower standard-wire is for the purpose of gauging the height of the lamp-flame in pure air. The presence of gas is then made known by the incandescence of the successive percentage-wires, the percentage of gas being indicated at once by the number of wires aglow. The successive percentage-wires indicate, respectively, 0.5, 1, 1.5, 2, 2.5 and 3 per cent. of gas in the air.

"In the calibration of the indicator an unbonneted Davy lamp, burning pure sperm oil, was used. The lamp was hung in a small wooden box 10 in. by 10 in. by 16 in., shown in Fig. 2. This box was connected below and above with a 10-in. pipe that formed a stack for the escape of the gas. The box was provided with a glass door at the front, through which the behavior of the lamp could be distinctly observed; the pipe was open at the bottom and top to permit the free upward passage of the air and gas. City gas was used, and this was introduced into the lower end of the pipe by the rubber tube leading from a gas-jet above. A damper in the pipe, just above the box, allowed the operator to accumulate the gas so as to obtain any desired gaseous condition of the air passing through the box in a constant stream; and it was also possible to maintain this constant condition of the passing current for any length of time, so as to enable a test to be made with the Shaw gas-machine located at one side of the box. A rubber tube connected this machine with a short brass pipe inserted in the back of the box and extending to its center, terminating at a point close to the gauze where the feed-air entered the lamp. By this means, it was possible to obtain an accurate test of the air that was feeding the lamps at any desired moment.

"Before placing an indicator in the lamp, experiments were made to ascertain the relation between the heights of the flame-cap and the percentages of gas causing the same, for the purpose of confirming the law formulated by Mr. William Gallo-

way, which says that the height of the cap varies with the cube of the percentage of gas present in the air. The height (h) in inches, for any percentage (J), was thus found to be expressed by the formulas,

Unbonneted Davy (sperm oil), $J = \sqrt[3]{36\ h}$ (Beard);
Bonneted Davy (sperm oil), $J = \sqrt[3]{70\ h}$ (Galloway).

"Having made these preliminary tests, an indicator-frame, with cross-wires of platinum arranged upon it at uniform distances apart, was placed in the lamp, and careful observations made to ascertain the relation of the height of incandescence of these wires to the percentage of gas in the air passing upward through the box. By this means, the heights of the several percentage-wires were determined. These observations and experiments, repeated many times and in different ways, revealed the discouraging fact that it was seemingly impracticable to attempt to insert a 0.5-per cent. wire, because its position was too close to the standard-wire used to gauge the flame. So close would this first percentage-wire be to the standard-wire that it required considerable care to set the flame so that it would incandesce the latter without causing both to glow. This difficulty, however, was successfully overcome later by using a slightly heavier iron wire in place of the platinum standard-wire. This iron wire had the effect of killing the heat in the immediate tip of the flame, with the result that each of the three lowest percentage-wires had to be raised. The lowest, or 0.5-per cent. wire, was raised the most; that next above, a less amount; and the upper one of the three, the least of all. The three upper percentage-wires were not disturbed in their position by the introduction of the iron wire for a standard-wire. The idea of thus killing the heat in the extreme tip of the flame was suggested by the principle of Sir Humphry Davy's wire-gauze.

"The iron standard-wire, however, proved a source of annoyance in the use of the indicator in strong gas. It would burn out in a short time, and required to be replaced by another wire. Further experiment showed that copper wire could not be used, probably owing to its high conductivity; aluminum, likewise, has a comparatively high conducting power, and, in addition, it

would not stand the heat of the flame. Finally, the use of a platinum wire of a somewhat lower gauge was found to meet every requirement.

"The advantage of the sight-indicator over the usual method of testing for gas may be briefly stated as follows: With the normal working-flame of a Davy or Clanny lamp, burning ordinary sperm or lard oil, it makes plainly visible within the lamp the slightest change in the gaseous condition of the mine-air, and indicates with great accuracy the exact percentage of gas present, in amounts varying from 0.5 to 3 per cent. Beyond this quantity the percentage of gas is estimated readily by the usual method of observing the height of the flame-cap, which is then clearly discernible. No time is lost in drawing down the flame when making a test, and the risk of losing the light is eliminated. The indicator, operating continuously and automatically, reveals the presence of unsuspected gas where a test by the usual method would be considered unnecessary; a possible accident may be thereby averted. Aside from the actual determination of the percentage of gas present, however, the chief advantage of such a device, in a mine, is its power to show constantly the fluctuation in the percentage of gas.

"The test is quickly made, as the wires respond promptly to the slightest change in the gaseous condition of the air. A change caused by the accidental setting open of a door, and the consequent derangement of the ventilating current, or the liberation of gas by a fall of roof in another portion of the mine, or an increased outflow of gas from old workings, caused by a sudden fall of the barometer, or even an increased quantity of inflammable dust in the air, will at once be shown by this silent monitor, as has been proved in several instances in the mine. It is already reported as having been the direct means of saving men's lives by indicating an unsuspected increase of gas where it was not thought necessary to stop to make a test. This will be understood when it is remembered that the device gives its indication with the normal working-flame, the bright incandescence of the wires being clearly seen, notwithstanding the brightness of the lamp-flame. This is a great advantage over the flame-cap method, in which it is necessary first to stop and lower the flame of the lamp to a mere glimmer before making the test. In doing this, there is incurred not only the

loss of time, but the risk of getting in the dark, which every miner rightly dreads.

"The appreciation of the value of this device for the purpose named is shown by the fact that special lamps have been designed for its use in England and also in France, because the mining laws of these countries forbid the use of an unbonneted Davy lamp in a gaseous mine, and it was thought best to design a special lamp in each case. Fig. 3 shows the sight-indicator in a Davy lamp; Fig. 4 shows it in a bonneted Clanny lamp, in which, however, its sensitiveness is somewhat lowered by the effect of the bonnet, which also hides some of the upper wires. Special forms of the Clanny and Mueseler lamps, the former shown in Fig. 5, having a 3.5-in. glass, and admitting the air below the flame, have recently been designed by me (Beard), which give good results with the indicator. The special form of English lamp of the Ashworth-Hepplewhite-Gray type, designed for the indicator, is shown in Fig. 6."

Postscript.—At the request of the Secretary of the Institute, the following information concerning the American and foreign patents of the Beard-Mackie indicator is here given:—

Country.	Number.	Date.
United States	692,885 722,555	Feb. 11, 1902. Mar. 10, 1903.
Canada	78,969 81,663	Jan. 13, 1903. June 30, 1903.
Great Britain	4,851 20,281	Feb. 26, 1902. Sept. 17, 1902.
Germany	171,891 185,268	Mar. 3, 1902. Sept. 20, 1902.
France	324,594	Sept. 18, 1902.
Belgium	165,669	Sept. 30, 1902.

Bibliography of Coal-Washing.

BY SAMUEL S. WYER, COLUMBUS, OHIO.

(Bethlehem Meeting, February, 1906.)

THE following abbreviations have been used in the text:

Am. Mfr. and Iron World. *American Manufacturer and Iron World*, Pittsburg, Pa.

Can. Min. Rev. *Canadian Mining Review*, Ottawa, Can.

Cass. Mag. *Cassier's Magazine*, New York, N. Y.

Col. Eng. *Colliery Engineer*, Scranton, Pa.

Col. Guard. *Colliery Guardian*, London, Eng.

Eng. and Min. Jour. *Engineering and Mining Journal*, New York, N. Y.

Eng. News. *Engineering News*, New York, N. Y.

Engr. *Engineering*, London, Eng.

J. I. and S. I. *Journal of the Iron and Steel Institute*, London, Eng.

Jour. W. Soc. of Engrs. *Journal Western Society of Engineers*, Chicago, Ill.

Lond. Eng. *The Engineer*, London, Eng.

Mines and Min. *Mines and Minerals*, Scranton, Pa.

North of Eng. I. M. E. *North of England Institute of Mining Engineers*, Newcastle-on-Tyne, England.

Oester. Zeitsch f. Berg u. Hüt. *Oesterreichische Zeitschrift für Berg-und Hüttenwesen*, Vienna, Austria.

Pract. Engr. *Practical Engineer*, London, Eng.

Proc. I. C. E. *Proceedings Institution of Civil Engineers*, London, Eng.

Trans. A. I. M. E. *Transactions American Institute of Mining Engineers*, New York, N. Y.

Trans. A. S. M. E. *Transactions American Society Mechanical Engineers*, New York, N. Y.

Trans. Fed. I. M. E. *Transactions Federated Institute Mining Engineers*, Newcastle-on-Tyne, England.

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Van Nostrand's Magazine, vol. 1, p. 280. Brief mention of coal-washing industry in France.

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J. I. and S. I., vol. 2, p. 680. Brief discussion of the progress of coal-washing in Belgium.

Proc. I. C. E., vol. 64, p. 426. Abstract of article on the Stutz coal-washer.

Trans. A. I. M. E., vol. 9, pp. 461 and 474. Elaborate description and illustrations of sizing and washing machinery, with discussion of principles involved and application of Rittenger's investigations.

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Proc. I. C. E., vol. 70, p. 106. Extensive description and elaborate drawings of an English coal-washing plant. Gives many valuable data.

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1885.

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Stahl und Eisen, vol. 7, p. 553. Description of a German coal-washing plant.

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Fulton, *Treatise on Coke*, p. 49. Beginning of elaborate discussion of coal-washing machinery. Gives description and illustrations of trough washer, Hartz jig, Berard's, Stutz-Stein's, Diescher's, Walburn-Swenson, Robinson's, Lührig and Dowlar's washers. Gives summary on page 88.

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Col. Guard., October 28. Illustration of washers.

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J. I. and S. I., vol. 2, p. 451. Short description of coal-washing plant.

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Kerr, *Practical Coal Mining*, p. 407. Gives classification of methods of coal-washing, with description and illustrations of several types of washers.

Lond. Eng., vol. 90, p. 98, July 27. Brief description of the Jeffrey coal-washing plants.

Mineral Industry, vol. 9, p. 731. Discussion of ore-dressing, with several references to coal-washing.

Smyth, *Coal and Coal Mining*, p. 292. Brief reference to washed coal.

Trans. I. M. E., vol. 21, p. 50. Brief reference to the importance of washed coal.

1901.

Col. Guard., August 16. Washing plant at a Belgian colliery.

Hughes, *Coal Mining*, p. 497. Brief discussion of coal-washing.

Jour. W. Soc. of Engrs., December. The washing of soft coals by the Lührig process

Mines and Min., September. Illustrated description of the coal-washing plant at Collinsville, Ind.

Peel, *Coal Mining*, p. 258. Brief reference to coal-washing.

1902.

Am. Mfr. and Iron World, vol. 70, p. 267. Brief discussion of the Maurice coal-washer. Gives summary of results.

Coal and Metal Miners' Handbook, p. 434. Discussion of coal-washing.

Coal and Metal Miners' Pocket-Book, p. 434. Discusses theory and operation of hydraulic classifiers, with a summary of the theory of jigging on page 439.

Col. Guard., vol. 84, p. 1119. Brief reference to value of coal-washing.

Col. Guard., vol. 84, p. 1393. Brief reference to the washing of coal.

Col. Guard., vol. 84, p. 1717. Brief reference to the washing of coal, giving description of Baum washer. Gives summary of results.

Engr., vol. 74, p. 239, August 22. Brief description of German coal-cleaning plants.

Engr., vol. 74, p. 441. Illustrations and description of the coal-washing plant at the Düsseldorf Exhibition. Very complete.

Eng. and Min. Jour., September 20. Discussion, with illustrations of coal-washing plant in Kansas.

Iron and Coal Trades Review, June 6. Illustrated description of the Campbell coal-washer.

Iron and Coal Trades Review, November 7. The screening and washing of coal in Germany. Serial, 1st part.

Mineral Industry, vol. 11, p. 657. Brief discussion of coal-washing.

Mines and Min., p. 36, August. Discussion of the principles of coal-washing. Gives mathematical deductions of formula.

Oester. Zeitsch. f. Berg u. Hüet., December 20. Detailed description of a modern coal-washing plant in Northern Austria.

Trans. A. I. M. E., vol. 32, p. 154. Brief description of coal-washing plant in Mexico.

Trans. I. M. E., vol. 23, p. 179. Description and illustrations of the Craig coal-washer, with discussion of its merits.

Trans. I. M. E., vol. 23, p. 435. Comprehensive article on the Campbell coal-washer. Gives illustrations, discussion of merits, and summary of results obtained.

1903.

Col. Guard., January 23. Illustrated description of a plant in Austria.

Col. Guard., vol. 85, p. 351. Brief reference to coal-washing.

Eng. and Min. Jour., May 9. A modern method of coal-washing. Gives drawings of the Campbell coal-washer.

Mines and Min., p. 456, May. Brief description of washing plant for Montana coal.

Mines and Min., p. 481, June. Description and illustrations of washing plant for handling anthracite culm. Drawings give general dimensions.

Mines and Min., June. Illustrates and describes the methods and machinery employed in handling and separating coal from culm banks.

Mines and Min., p. 212, December. Description and illustrations of the Stewart coal-washing system. Gives summary of costs.

Mines and Min., p. 215, December. Brief quotation from U. S. Geological Survey Bulletin on "The Manufacture of Coke."

Mines and Min., p. 228, December. Description and illustration of plant for washing Montana coal.

Richards, *Ore Dressing*. Both volumes of the above treatise are replete with data on sizing, jigging, and classification of ores.

1904.

Economical Burning of Coal Without Smoke, p. 28. Publication of the Peabody Coal Company.

Mines and Min., p. 371, March. Description and good illustrations of coal-washing plant in Indian Territory.

Mines and Min., December. Illustrated description of washing plant of the Colorado Fuel and Iron Co.

Mines and Min., December. Description of various trough and jigging washers. Gives tables and costs.

Mines and Min., December. Description and illustration of washing plant using Robinson and Stewart washers.

1905.

Cocklin, *Coal Mining*, p. 376. Several illustrations of coal-washers.

Eng. and Min. Jour., vol. 80, p. 867. Discussion of shaking screens for coal-washing.

Fulton, *Coke*, p. 56. Thorough discussion, with numerous illustrations of washers. Gives results of various tests. This is the best treatment of the subject yet published.

Mines and Min., vol. 26, p. 87. Brief description of German coal-washing plant.

Screens for Sizing.

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(Bethlehem Meeting, February, 1906.)

ACCURATE ore-sizing with screens is drawing attention to certain details that now, more than ever before, require attention. There are many tests that must be preceded by careful sizing. The assayer often could profit by a more careful selection of screens. In milling, more attention is given the slimes produced. Cement-manufacturers are pulverizing and screening finer than they formerly did, and others are reducing materials to fine sizes at high cost, with screens used to govern the fineness of the pulverized product. Fine screens are expensive, easily injured by misuse, are carried in stock sparingly by most dealers, and are handled often by those who know little about them except their designated number.

More attention, on the whole, needs be given the actual size of the mesh-openings, and less emphasis placed upon any custom or conventionality in numbering. An effort to classify wire screens, in this respect, leads at once into computations, where one is confronted with details of sizes and standards, of different wires, different gauges, and different customs. There is always uncertainty about the exact working-size of a screen until one gives the matter special attention, and devotes more time to it than its simplicity seems to demand. To know only the number is not sufficient for all uses. Often, in sets, superfluous sizes are used; and sizes are omitted that are of critical importance.

In wire screens, the use of different standard sizes of wire makes important differences in dimensions that must be taken into account. The dealer usually grades screens by the number of openings per linear dimensions, regardless of the size of the openings or of the wire. Certain 100-mesh screens in use are found, by measurement, to be actually coarser than others of 90-mesh, a result which is due, of course, to the use of larger

wire in the latter than in the former, while in number the openings remain correct.

In arranging a graded series of screens, following any mathematical law or the line of any curve, the tables which follow will be found of use. The working-spaces of standard screens of the different sizes can be compared in these tables; and the modulations of these spaces that result from varying the wire to produce a different weight of screen, or a different strength, can be seen and chosen from the many combinations possible.

A system of numbering screens by their space-openings would be most desirable could it become general. Numbering from 1 to 1,000 in thousandths of an inch, as has been adopted by certain English manufacturers, for the measurement of wire, or from 1 to 250 in hundredths of a millimeter, were the metric system to be employed, would have many advantages. It is well, in following one's own purpose, to give to the screens such numbers as designate their working-sizes, and to regard them by the dimensions of their spaces. By doing this more generally, a better expression of the actual size of screened material would be had than is now common.

The possibility of classifying materials in currents of water, or of air, at different velocities, makes the use of screens, and, fortunately, the finer ones, often unnecessary. But classification by difference in size alone, with no specific gravity condition to enter, requires actual screening. For this need, there are available, and on the market, punched screens and wire screens, of various metals, and standard sizes. The punched screens are used extensively in gold-milling, and are regarded by many as superior to other kinds for this purpose. Such screens are made with punched openings, both round and square; and many are punched with the long and narrow needle slot, cut either along horizontal lines or on a diagonal. Some of the screens are punched with a burr purposely left around the edge of the slots, and others with indented slots.

The thickness of a punched metal usually should be between a half and a quarter of the diameter of the hole; but this requisite varies also with the size, and with the kind and the quality of metal. However desirable extreme thinness of the metal would be, causing screens to work freely in the fine sizes, the strength, notwithstanding, must be maintained, and allow-

ance made for the wear of the sheet-metal. Screens are made thicker, relatively to the size, therefore, as they are made to do finer work, since there must be adequate working-area and strength of screen to support this.

Screens made from drawn wire of different metals are strong. Injury most often comes to these from the displacement of wires; and this quality, with the high first cost, often gives reason for using other kinds. Very fine screens are made, desirably, of wire cloth. For many uses, the wire screen has no satisfactory substitute.

The shape and area of the apertures through which the crushed material must pass is the important quality, if the screen be used for accurate work. Different shapes of screen-openings act differently upon particles of different shapes. Tabular particles can pass the slot-screens when they would be held back by square or round openings of equal space; and the passage of needle-like particles must depend, in a measure, upon the kind and violence of the motion and the thickness of the sheet-metal. Different minerals and rocks break into different shapes of particles; and the methods of pulverizing influence this. Abrasion tends to destroy any extreme irregularity of shape, and so, for the most part, a crushed substance like ore breaks into particles of no very great disparity of axial dimensions.

If one might neglect these differences in the shape (which result from cleavage and crystallization, and which arise from the different action of the crusher upon particles that are brittle, elastic or malleable), the mean condition of dimensions, or the average shape of particles, would be spherical. This shape cannot be fairly neglected; but in the grading of screens it is well to consider their work as if upon either spherical or cubical particles; and where difference in the shape is taken into consideration, one then may use the sphere or the cube as a basis in making the necessary modification. If uniform rock tends to assume certain forms in the large pieces, it is likely to do this also in the small sizes, and the relation between diameter and mass then holds through all sizes. Ores, however, being composed of mixed minerals, liberate crystals or minerals of different shapes upon crushing, and no relation holds that can be predicted definitely. Allowance usually must be

made for the widest difference in size that could be probable among these particles, as they escape from the rough action of the crusher.

This difference will be found greater than is desired, but it must not in any way discourage close work with screens. Brunton[1] finds with a single mineral (pyrite), particles passing the screen which are nearly three times the size (mass) of the theoretical cube that could pass. On the other hand, with a mineral like quartz, pieces commonly are taken from screens which are only one-third and one-fourth the weight of the theoretical cube.

Thus, there is an overlapping in the actual weight of particles from one size of screen to the next, in a closely graded set. This lays emphasis upon the motion that is given the screen to force the ore through, upon the length of time allowed the particles to find a position to pass, and upon the mass of material on the screen obstructing the freedom of motion, with other conditions that modify the mechanical action, all of which, if not made uniform, would have much influence upon the results obtained with screens uniform in size.

In the fine sizes, moisture in the ore has influence. The different frictional resistance of fine particles, too, affects their behavior. Their contacts, with increased surface and smaller masses, influenced by an increase of their appreciable cohesive force, affect them. The diminished force of gravity and momentum acting upon these smaller masses having greater proportionate surfaces, makes the departure from the behavior of the theoretical cube different for different sizes.

The effects of the size of the screen are distinct, however, upon the average size of a product, or upon products typified, in single sizes. Deviations from such a typical condition are characteristic of the ore, and are functions of this, as well as of the size to which the ore is crushed. Screens are gauged as accurately, therefore, as though the products were strictly uniform.

The slot-screen, in this regard, has only one dimension to be considered, the width of the slot. The length of the slot is limited chiefly by the strength desired, and for ore-sizing, this

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[1] *Trans.*, xxv., 832 (1895).

length is sufficient to be disregarded. The square-punched or the woven wire screen must be rated according to the space between wires, or the diameter of the largest possible inscribed circle in the rectangular opening. The round-punched screen is measured by the diameter of the openings.

The thickness of a punched sheet, also, has something to do with the size of the irregularly-shaped particles that actually pass; but thickness, necessarily, increases the tendency of a screen to clog, and any difference in size that can arise from difference in thickness, if a screen is suitable to use at all, is small compared with other factors.

Microscopic examination shows that most fine cloth-screens are well made; and that the wire from which they are woven is gauged evenly by some standard intended by the manufacturer. The benefit of this accuracy, however, may not always fall upon the consumer, who finds difficulty, perhaps, in obtaining the exact size desired without delay, or without danger of specifying sizes difficult to produce.

A large number of screens of many kinds have been measured by projecting parallel rays of light through them and through the lenses of a stereopticon to a wall upon which they could be measured. After testing the lenses of a stereopticon, measurement can be made conveniently in this way with much accuracy. A large field is exposed to view; and aberration of light is avoided by fastening pieces of the fine screens upon ground-glass slides for their projection. Any irregularity is detected at once by the eye, and the finest wire is measured accurately with a drawing-scale.

The result of the examination of these screens shows the need of care by all who are using screens for accurate work. One must be certain not only that the number of apertures per linear inch is correct, but that the size of wire, if uniform, is of the size expected; and one must see what effect this size has upon the diameter of the available aperture.

In the following tables are given the space-openings of screens of different sizes, when made correctly of standard wire. The calculation upon which the tables are based refers to the size of spherical particles which screens would permit to pass. Different standard-gauges are shown, as are required to cover the needs of the prevailing manufacturing customs.

Whereas, iron or steel usually is measured by the Worcester (Washburn and Moen or the Roebling) gauge, copper is measured by the American (Brown and Sharpe) gauge, and brass wire by the Old English, or London gauge, or by the Birmingham (or Stub's) gauge in the coarser sizes.

The tables can be used to select from the standard sizes such screens as are required in any graded series. Within a certain limit, it is justifiable to vary the wire in ordering, so as to obtain the exact sizes wanted. For this purpose, screens of different weight, of different sizes of wire, are shown.

The letter *d* represents the screens used by Prof. De Kalb in his tests and graphic records.[2]

The letter *f* shows the screens which most nearly approach a series based upon quarters, eighths and sixteenths of inches, as mentioned below.

The letter *h* represents screen-sizes that correspond to those described by W. S. Hutchinson[3] and used in connection with Prof. Richards' work on Ore Dressing.

The letter *p* shows those sizes that closely approach the space of certain standard needle-slot screens which are made.

The letter *r*, in the reference column, shows sizes closely approaching those used by Rittinger, obtained by the use of a size of wire recommended in a table published by the *School of Mines Quarterly*.[4]

The letter *s* shows sizes used by Brunton in his work upon ore-sampling.[5]

The letter *t* shows the sizes of all those desirable screens, in each table, which approach most nearly the sizes represented in Table I.

The tables will be found arranged in the order of decreasing sizes resulting from the use of heavier wire, within limits. Sizes will be found approaching closely any sizes required in a graded series.

A series, to be consistent, often should follow, in geometrical progression, a constant fractional reduction from size to size. Thus, a series of 2, 1, $\frac{1}{2}$, $\frac{1}{4}$, $\frac{1}{8}$, $\frac{1}{16}$, $\frac{1}{32}$, $\frac{1}{64}$, $\frac{1}{128}$, $\frac{1}{256}$ and $\frac{1}{512}$ in., or, expressed in decimals, 2, 1, 0.5, 0.25, 0.125, 0.0625,

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2 *Trans.*, xxviii., 484 (1898).

3 *Trans.*, xxxv., 258 (1905).

4 *School of Mines Quarterly*, vol. ii., p. 216 (May, 1881).

5 *Trans.*, xxv., 827 (1895).

0.0312, 0.0156, 0.0078, 0.0039 and 0.0020 in., is satisfactory for general use. The nearest approach to such a series of screens as this, is shown by the letter *f* in the tables.

A desirable graded series of screens is shown by the space openings represented in Table I. These follow the lines of such a curve as has been used by Rittinger, and they are extended in the tabulation, here, in order to cover the extremely fine sizes. The nearest approach to these desirable space openings, that can be obtained by the use of standard wire, is here represented by the mesh of the screen and the number of the wire.

TABLE I.—*A Graded Series of Screens.*

Openings Desired.		Meshes* to the Inch.	Size of Brass Wire. Old English or London Gauge.
Millimeters.	Inches.		
64.0	2.56	3″	000
45.2	1.80	2⅛″	0
32.0	1.28	1½″	3
22.6	0.890	1⅛″	4
16.0	0.630	¾″	11
11.3	0.445	⅝″	7
8.0	0.315	2½	14
5.6	0.222	3½	16
4.0	0.157	4	13
		4½	16
		5	19
2.8	0.111	7	21
2.0	0.079	7	16
		9	21
1.41	0.056	12	23
		14	29
1.00	0.039	16	25
		18	28
		20	32
		22	36
0.70	0.028	18	23
		20	25
		22	26
		24	30
		26	33

Openings Desired.		Meshes to the Inch.	Size of Brass Wire. Old English or London Gauge.
Millimeters.	Inches.		
0.50	0.020	26	27
		28	29
		30	30
		35	35
0.35	0.014	35	30
		40	32
		50	38
0.25	0.0098	50	34
		55	35
0.175	0.0069	74	37
		80	38
0.125	0.0049	90	38
		100	39
		120	43
0.087	0.0034	150	44
		160	45
0.062	0.0024	160	43
		180	44
		190	45
		200	45½
0.043	0.0017	180	43
		190	43
		200	44
0.031	0.0012	190	42
0.022	0.0009	200	42

* Where the sign of inches (″) is expressed, the number signifies "inches to the mesh."

The needle-slot screens do not conform to screens of wire-cloth in their designating numbers. In punched screens, the size of the opening is incidental to nothing else. Coarse sizes can be obtained of any space-opening desired in simple fractions

of inches or millimeters. Certain finer sizes, which are represented by smaller fractions, are shown in Table II.

TABLE II.—*Openings in Certain Punched Screens.*

No.	Mesh.	Width of Slot.		No.	Mesh.	Width of Slot.	
		Inches.	Millimeters			Inches.	Millimeters.
1	12	0.058	1.47	8	35	0.022	0.56
2	14	0.049	1.24	9	40	0.020	0.51
3	16	0.042	1.07	10	50	0.018	0.46
4	18	0.035	0.89	11	55	0.0165	0.42
5	20	0.029	0.74	12	60	0.015	0.38
6	25	0.027	0.69	13	70	0.0135	0.34
7	30	0.024	0.61				

In Tables III. to VI., inclusive, the following references are used:

d. Represents the size of screens used by De Kalb.[6]

f. Represents the nearest approach to sizes of a set graded as follows:—1″, ½″, ¼″, etc. (see p. 270).

h. Shows screens which have been used by Richards as described by Hutchinson.[7]

p. Represents sizes corresponding most nearly to the punched screens given in Table II.

s. Represents the screens used by Brunton.[8]

t. Represents all sizes shown in Table I.

r. Shows Rittinger screens, elsewhere advised.[9]

In the column representing the screen numbers, when the sign of inches (″) is expressed, the number is understood to imply "inches to the mesh." When no sign of inches is shown, the number signifies the number of "meshes to the inch." Thus, a ¾″ screen is also a 1⅓-mesh screen. Mesh-dimensions, in all cases, are represented as distances between wire-centers. The working size of screens, or the opening, or aperture, is the clear "space" between wires. Decimal values beyond the number here retained have been used where required for deducing the corresponding values of the other columns. The number of places shown is a convenient number, and is regarded as consistent with the possibilities in practice.

[6] *Trans.*, xxviii., 484 (1898).
[7] *Trans.*, xxxv., 258 (1905).
[8] *Trans.*, xxv., 827 (1895).
[9] *School of Mines Quarterly*, vol. ii., p. 216 (1880–1).

TABLE III.—*Sizes of Wire Screens (usually iron or steel) of the Worcester (Washburn & Moen) (Roebling) Standard.*

Mesh.	Wire. W. & M.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
Inches to the mesh.					
3″	0	0.307	2.69	68.3	
3″	00	0.331	2.67	67.8	r
3″	000	0.362	2.64	67.1	t
2½″	1	0.283	2.22	56.4	
2½″	0	0.307	2.19	55.6	
2½″	00	0.331	2.17	55.1	f
2⅛″	0	0 307	1.82	46.2	rt
1½″	3	0.244	1 26	32.0	t
1½″	2	0.263	1 24	31 5	r
1½″	1	0.283	1.22	31.0	
1⅛″	4	0 225	0.900	22.9	rtf
13/16″	7	0.177	0 635	16.1	r
1″	15	0.072	0.928	23.6	
1″	14	0.080	0.920	23.4	
1″	13	0.092	0.908	23.1	
1″	12	0.105	0.895	22.7	
1″	11	0.120	0.880	22.3	
1″	10	0.135	0.865	22.0	
1″	9	0.148	0.852	21.6	
1″	8	0.162	0.838	21.3	
1″	7	0.177	0.823	20.9	
1″	6	0.192	0.808	20.5	
1″	5	0.207	0.793	20.1	
1″	4	0 225	0.775	19.7	
1″	3	0.244	0.756	19.2	s
¾″	16	0.063	0.687	17.4	
¾″	15	0.072	0.678	17.2	
¾″	14	0.080	0.670	17.0	
¾″	13	0.092	0.658	16.7	
¾″	12	0.105	0.645	16.4	
¾″	11	0.120	0.630	16.0	t
¾″	10	0.135	0.615	15.6	
¾″	9	0.148	0.602	15.3	
¾″	8	0.162	0.588	14.9	
¾″	7	0.177	0.573	14.5	
¾″	6	0.192	0.558	14.2	
¾″	5	0.207	0.543	13.8	
¾″	4	0.225	0.525	13.3	
⅝″	17	0.054	0.571	14.5	
⅝″	16	0.063	0.562	14.3	
⅝″	15	0.072	0.553	14.0	
⅝″	14	0 080	0.545	13.8	
⅝″	13	0.092	0.533	13.5	
⅝″	12	0.105	0.520	13.2	
⅝″	11	0.120	0.505	12.8	f
⅝″	10	0.135	0.490	12.4	
⅝″	9	0.148	0.477	12.1	
⅝″	8	0.162	0.463	11.8	r
⅝″	7	0.177	0.448	11.4	t
⅝″	6	0.192	0.433	11.0	
⅝″	5	0.207	0.418	10.6	
Meshes to the inch.					
2	18	0.047	0.453	11.5	
2	17	0 054	0.446	11.3	
2	16	0.063	0.437	11.1	
2	15	0.072	0.428	10.9	
2	14	0.080	0.420	10.7	
2	13	0.092	0.408	10.4	
2	12	0.105	0.395	10.0	
2	11	0.120	0.380	9.65	
2	10	0.135	0.365	9.27	
2	9	0.148	0.352	8.94	
2	8	0.162	0 338	8.59	s
2	7	0.177	0.323	8.20	
2	6	0.192	0.308	7.82	
2½	19	0.041	0.359	9.12	
2½	18	0.047	0.353	8.97	
2½	17	0.054	0.346	8.79	
2½	16	0.063	0.337	8.56	
2½	15	0.072	0.328	8.33	
2½	14	0.080	0.320	8.13	t
2½	13	0.092	0.308	7.82	
2½	12	0.105	0.295	7.49	r

Mesh.	Wire. W. & M.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
Meshes to the inch.					
2½	11	0.120	0.280	7.11	
2½	10	0.135	0.265	6.73	
2½	9	0.148	0.252	6.40	
2½	8	0 162	0.238	6.05	
3	20	0 035	0.298	7.57	
3	19	0 041	0.292	7.42	
3	18	0.047	0.286	7.26	
3	17	0.054	0.279	7.09	
3	16	0.063	0.270	6.86	
3	15	0.072	0.261	6.63	
3	14	0.080	0.253	6.43	f
3	13	0.092	0.242	6.15	
3	12	0.105	0.228	5.79	
3	11	0 120	0.213	5.41	
3	10	0.135	0.198	5.03	s
3½	21	0.032	0.253	6.43	
3½	20	0.035	0.250	6.35	
3½	19	0 041	0.244	6.20	
3½	18	0.047	0.238	6.05	
3½	17	0 054	0.231	5.87	
3½	16	0 063	0.222	5.64	t
3½	15	0 072	0.213	5.41	
3½	14	0 080	0.205	5.21	r
3½	13	0 092	0.193	4.90	
3½	12	0.105	0 180	4.57	
3½	11	0.120	0.165	4.19	
4	22	0.028	0.222	5.64	
4	21	0.032	0.218	5.54	
4	20	0.035	0 215	5.46	
4	19	0.041	0.209	5.31	
4	18	0.047	0 203	5 16	
4	17	0.054	0.196	4.98	
4	16	0.063	0.187	4.75	
4	15	0.072	0.178	4.52	
4	14	0 080	0.170	4.32	
4	13	0.092	0.158	4.01	t
4	12	0.105	0.145	3.68	s
4	11	0.120	0.130	3.30	
4½	20	0.035	0.187	4.75	
4½	19	0.041	0.181	4.60	
4½	18	0.047	0.175	4.45	
4½	17	0.054	0.168	4.27	
4½	16	0.063	0.159	4.04	
4½	15	0.072	0.150	3.81	r
4½	14	0.080	0.142	3.61	
4½	13	0.092	0.130	3.30	
4½	12	0.105	0 117	2.97	
5	23	0.025	0.175	4.44	
5	22	0.028	0.172	4.37	
5	21	0.032	0.168	4.27	
5	20	0.035	0.165	4.19	
5	19	0.041	0.159	4.04	
5	18	0.047	0.153	3.89	
5	17	0.054	0.146	3.71	
5	16	0.063	0.137	3.48	
5	15	0.072	0.128	3.25	
5	14	0.080	0.120	3.05	
5	13	0.092	0.108	2.74	s
6	24	0.023	0.144	3.66	
6	23	0.025	0.142	3.61	
6	22	0.028	0.139	3.53	
6	21	0.032	0.135	3.43	
6	20	0.035	0.132	3.35	
6	19	0.041	0.126	3.20	f
6	18	0.047	0.120	3.05	
6	17	0.054	0.113	2.87	
6	16	0.063	0.104	2.64	r
6	15	0.072	0.095	2.40	
6	14	0 080	0.087	2.20	s
7	26	0.018	0.125	3.17	
7	25	0.020	0.123	3.12	
7	24	0.023	0.120	3.05	

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *p.* Nearest size to punched screens, Table II. *s*, Brunton's screens. *t.* Sizes in Table I. *r.* Rittinger's screens,

Table III.—*Continued.*

Mesh. Meshes to the inch.	Wire. W. & M.	Wire. Diameter in Inches	Space. Inches.	Space. Milli-meters.	Reference.	Mesh. Meshes to the inch.	Wire. W. & M.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
7	23	0.025	0.118	3.00		14	20	0.035	0.036	0.92	s
7	22	0.028	0.115	2.92		16	35	0.0095	0.053	1.35	
7	21	0.032	0.111	2.82	t	16	34	0.010	0.052	1.33	
7	20	0.035	0.108	2.74		16	33	0.011	0.051	1.31	
7	19	0.041	0.102	2.59		16	32	0.013	0.050	1.26	
7	18	0.047	0.096	2.44		16	31	0.0135	0.049	1.24	
7	17	0.054	0.089	2.26		16	30	0.014	0.048	1.23	
7	16	0.063	0.080	2.03		16	29	0.015	0.047	1.21	
7	15	0.072	0.071	1.80	s	16	28	0.016	0.046	1.18	
8	27	0.017	0.108	2.74		16	27	0.017	0.045	1.16	
8	26	0.018	0.107	2.72		16	26	0.018	0.044	1.13	
8	25	0.020	0.105	2.67		16	25	0.020	0.042	1.08	pr
8	24	0.023	0.102	2.59		16	24	0.023	0.039	1.00	
8	23	0.025	0.100	2.54		16	23	0.025	0.037	0.95	
8	22	0.028	0.097	2.46		16	22	0.028	0.034	0.88	s
8	21	0.032	0.093	2.36		16	21	0.032	0.030	0.77	
8	20	0.035	0.090	2.29		18	36	0.009	0.047	1.18	
8	19	0.041	0.084	2.13		18	35	0.0095	0.046	1.17	
8	18	0.047	0.078	1.98	r	18	34	0.010	0.045	1.16	
8	17	0.054	0.071	1.80		18	33	0.011	0.044	1.13	
8	16	0.063	0.062	1.57	s	18	32	0.013	0.043	1.08	
9	28	0.016	0.095	2.42		18	31	0.0135	0.042	1.07	
9	27	0.017	0.094	2.39		18	30	0.014	0.041	1.05	
9	26	0.018	0.093	2.36		18	29	0.015	0.040	1.03	
9	25	0.020	0.091	2.31		18	28	0.016	0.039	1.00	t
9	24	0.023	0.088	2.24		18	27	0.017	0.038	0.98	
9	23	0.025	0.086	2.19		18	26	0.018	0.037	0.95	
9	22	0.028	0.083	2.11		18	25	0.020	0.035	0.90	p
9	21	0.032	0.079	2.01	t	18	24	0.023	0.032	0.82	
9	20	0.035	0.076	1.93		18	23	0.025	0.030	0.77	s
9	19	0.041	0.070	1.78		18	22	0.028	0.027	0.69	
9	18	0.047	0.064	1.63		20	35	0.0095	0.041	1.03	
9	17	0.054	0.057	1.45	s	20	34	0.010	0.040	1.02	
10	29	0.015	0.085	2.17		20	33	0.011	0.039	0.99	
10	28	0.016	0.084	2.13		20	32	0.013	0.038	0.94	
10	27	0.017	0.083	2.11		20	31	0.0135	0.037	0.93	
10	26	0.018	0.082	2.08		20	30	0.014	0.036	0.91	
10	25	0.020	0.080	2.03		20	29	0.015	0.035	0.89	
10	24	0.023	0.077	1.96		20	28	0.016	0.034	0.86	
10	23	0.025	0.075	1.90		20	27	0.017	0.033	0.84	
10	22	0.028	0.072	1.83		20	26	0.018	0.032	0.81	
10	21	0.032	0.068	1.73		20	25	0.020	0.030	0.76	p
10	20	0.035	0.065	1.65		20	24	0.023	0.027	0.69	s
10	19	0.041	0.059	1.50		20	23	0.025	0.025	0.63	
10	18	0.047	0.053	1.35	s	22	35	0.0095	0.036	0.91	
10	17	0.054	0.046	1.17		22	34	0.010	0.035	0.89	
12	30	0.014	0.069	1.75		22	33	0.011	0.034	0.86	
12	29	0.015	0.068	1.73		22	32	0.013	0.033	0.84	
12	28	0.016	0.067	1.70		22	31	0.0135	0.032	0.81	
12	27	0.017	0.066	1.68		22	30	0.014	0.031	0.79	f
12	26	0.018	0.065	1.65		22	29	0.015	0.030	0.76	
12	25	0.020	0.063	1.60	f	22	28	0.016	0.029	0.74	
12	24	0.023	0.060	1.52		22	27	0.017	0.028	0.71	
12	23	0.025	0.058	1.47	p	22	26	0.018	0.027	0.69	
12	22	0.028	0.055	1.40	rt	22	25	0.020	0.025	0.64	
12	21	0.032	0.051	1.30		22	24	0.023	0.022	0.56	s
12	20	0.035	0.048	1.22		24	36	0.009	0.033	0.83	
12	19	0.041	0.042	1.07	s	24	35	0.0095	0.032	0.82	
14	34	0.010	0.061	1.56		24	34	0.010	0.031	0.80	
14	33	0.011	0.060	1.53		24	33	0.011	0.030	0.78	
14	32	0.013	0.059	1.48		24	32	0.013	0.029	0.73	
14	31	0.0135	0.058	1.47		24	31	0.0135	0.028	0.71	
14	30	0.014	0.057	1.46		24	30	0.014	0.027	0.70	t
14	29	0.015	0.056	1.43		24	29	0.015	0.026	0.68	p
14	28	0.016	0.055	1.41		24	28	0.016	0.025	0.65	r
14	27	0.017	0.054	1.38		24	27	0.017	0.024	0.62	
14	26	0.018	0.053	1.36		24	26	0.018	0.023	0.60	s
14	25	0.020	0.051	1.31		24	25	0.020	0.022	0.55	
14	24	0.023	0.048	1.23	p	26	36	0.009	0.030	0.75	
14	23	0.025	0.046	1.18		26	35	0.0095	0.029	0.73	
14	22	0.028	0.043	1.10		26	34	0.010	0.028	0.72	
14	21	0.032	0.039	1.00		26	33	0.011	0.027	0.70	

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *p.* Nearest size to punched screens, Table II. *s.* Brunton's screens. *t.* Sizes in Table I. *r*, Rittinger's screens.

TABLE III.—*Concluded.*

Mesh. (Meshes to the inch.)	Wire. W. & M.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
26	32	0.013	0.026	0.64	
26	31	0.0135	0.025	0.63	
26	30	0.014	0.024	0.62	p
26	29	0.015	0.023	0.59	
26	28	0.016	0.022	0.57	
26	27	0.017	0.021	0.54	
26	26	0.018	0.020	0.52	
28	36	0 009	0.027	0.68	
28	35	0 0095	0.026	0.67	
28	34	0.010	0.025	0.65	
28	33	0.011	0.024	0.63	
28	32	0.013	0 023	0.58	
28	31	0.0135	0.022	0.56	
28	30	0.014	0.021	0.55	
28	29	0 015	0.020	0.53	
28	28	0.016	0.019	0.50	
30	35	0.0095	0.024	0.60	
30	34	0.010	0.023	0.59	
30	33	0.011	0 022	0.57	p
30	32	0.013	0.021	0.52	
30	31	0.0135	0 020	0.50	pt
30	30	0.014	0.019	0.49	
30	29	0.015	0.018	0.46	
30	28	0.016	0.017	0 44	s
30	27	0.017	0.016	0.41	
35	36	0.009	0.020	0.50	
35	35	0.0095	0.019	0.48	
35	34	0.010	0.018	0.47	p
35	33	0.011	0.017	0.44	
35	32	0.013	0.016	0.39	f
35	31	0.0135	0.015	0.38	r
35	30	0.014	0.014	0.37	
35	29	0 015	0 013	0.34	
40	37	0.0085	0.0165	0.42	p
40	36	0.009	0.016	0.41	
40	35	0.0095	0.0155	0.39	
40	34	0.010	0.015	0.38	p
40	33	0 011	0.014	0.36	pt
40	32	0.013	0.012	0.30	s
40	31	0.0135	0.0115	0.29	
40	30	0 014	0.011	0.28	
45	37	0.0085	0.014	0.35	
45	36	0.009	0.013	0.33	
45	35	0.0095	0.0125	0.32	
45	34	0.010	0.012	0.30	
45	33	0.011	0.011	0.28	
45	32	0.013	0.009	0.23	

Mesh. (Meshes to the inch.)	Wire. W. & M.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
50	37	0.0085	0.0115	0.29	
50	36	0.009	0.011	0.28	
50	35	0.0095	0.0105	0 27	s
50	34	0.010	0.010	0.25	rt
50	33	0.011	0.009	0.23	
50	32	0.013	0.007	0.18	
55	38	0.008	0.010	0.25	
55	37	0.0085	0.0095	0.24	
55	36	0.009	0.009	0.23	
55	35	0 0095	0 0085	0 22	
55	34	0.010	0.008	0.20	
60	39	0.0075	0.009	0.23	
60	38	0.008	0 0085	0.22	
60	37	0.0085	0.008	0 20	f
60	36	0.009	0.0075	0.19	
60	35	0.0095	0.007	0.18	
64	40	0.007	0 0085	0.22	
64	39	0.0075	0.008	0 20	
64	38	0.008	0.0075	0.19	
64	37	0.0085	0.007	0.18	
64	36	0.009	0 0065	0.17	
70	40	0.007	0.007	0.18	
70	39	0.0075	0.0065	0.17	t
70	38	0.008	0.006	0.15	r
70	37	0.0085	0 0055	0.14	
70	36	0.009	0 005	0.13	
74	41	0.0066	0.007	0.18	
74	40	0.0070	0.0065	0.17	
74	39	0.0075	0 006	0.15	
74	38	0.0080	0.0055	0.14	
74	37	0.0085	0 005	0.13	
80	42	0.0062	0.0065	0.16	
80	41	0.0066	0 006	0 15	
80	40	0.0070	0 0055	0.14	
80	39	0.0075	0.005	0.13	t
80	38	0.0080	0.0045	0 11	
80	37	0.0085	0 004	0 10	
90	42	0.0062	0 005	0.12	
90	41	0.0066	0.0045	0.11	
90	40	0.0070	0.004	0.10	f
90	39	0.0075	0.0035	0.09	
90	38	0.0080	0.003	0 08	
100	42	0.0062	0 004	0.10	
100	41	0.0066	0.0035	0.09	t
100	40	0.0070	0.003	0.08	
100	39	0.0075	0.0025	0.06	t

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *p*. Nearest size to punched screens, Table II. *s*. Brunton's screens. *t*. Sizes in Table I. *r* Rittinger's screens.

TABLE IV.—*Sizes of Wire Screens (usually brass) of the Old English or London Gauge.*

	Mesh.	Wire. Old English.	Wire. Diameter in Inches.	Space. Inches.	Space. Millimeters.	Reference.		Mesh.	Wire. Old English.	Wire. Diameter in Inches.	Space. Inches.	Space. Millimeters.	Reference.
Inches to the mesh.	3″	0	0 340	2 66	67.6		Meshes to the inch.	3	20	0 035	0.298	7.57	
	3″	00	0.380	2.62	66.5	r		3	19	0.040	0.293	7 44	
	3″	000	0.425	2.57	65.3	t		3	18	0.049	0.284	7.21	
	2½″	1	0.300	2.20	55.8			3	17	0.058	0.275	6.98	
	2½″	0	0.340	2.16	54.9			3	16	0.065	0.268	6.81	
	2½″	00	0.380	2.12	53.8	f		3	15	0.072	0.261	6.63	
	2⅛″	0	0.340	1.78	45.2	rt		3	14	0.083	0.250	6.35	f
	1½″	3	0.259	1 24	31.5	rt		3	13	0.095	0.238	6.05	
	1½″	2	0.284	1.22	31.0	r		3	12	0 109	0.224	5.69	
	1½″	1	0.300	1 20	30.5			3	11	0.120	0.213	5.41	
	1⅛″	4	0.238	0.89	22.8	rt		3	10	0.134	0.199	5.05	
	1″	15	0.072	0.928	23.6	f		3½	20	0.035	0.250	6.35	
	1″	14	0 083	0.917	23.3			3½	19	0.040	0.245	6.22	
	1″	13	0.095	0.905	23.0			3½	18	0.049	0.236	5.99	
	1″	12	0.109	0.891	22.6	d		3½	17	0.058	0.227	5.77	
	1″	11	0.120	0 880	22.3			3½	16	0.065	0.220	5.59	t
	1″	10	0.134	0.866	22.0			3½	15	0.072	0.213	5.41	
	1″	9	0.148	0.852	21.6			3½	14	0.083	0.202	5.13	r
	1″	8	0.165	0.835	21.2			3½	13	0.095	0.190	4.83	
	1″	7	0.180	0.820	20.8			3½	12	0.109	0.176	4.47	
	1″	6	0.203	0.797	20.2			3½	11	0.120	0.165	4.19	
	1″	5	0.220	0.780	19 8			4	21	0.0315	0.218	5.54	
	1′	4	0.238	0.762	19.3			4	20	0.035	0.215	5.46	
	1″	3	0.259	0.741	18 8			4	19	0.040	0.210	5.33	
	¾″	16	0.065	0.685	17.4			4	18	0.049	0.201	5.11	d
	¾″	15	0.072	0.678	17.2			4	17	0.058	0.192	4.88	
	¾″	14	0.083	0.667	16.9			4	16	0.065	0.185	4.70	
	¾″	13	0.095	0 655	16.6	d		4	15	0.072	0.178	4.52	
	¾″	12	0.109	0 641	16 3			4	14	0.083	0.167	4.24	
	¾″	11	0.120	0 630	16.0	t		4	13	0.095	0.155	3.94	t
	¾″	10	0.134	0 616	15.6			4	12	0.109	0.141	3.58	
	¾″	9	0.148	0.602	15.3			4	11	0.120	0.130	3.30	
	¾″	8	0.165	0.585	14.9			4½	21	0.0315	0.191	4.85	
	¾″	7	0.180	0.570	14.5			4½	20	0.035	0.187	4.75	
	¾″	6	0.203	0.547	13.9			4½	19	0.040	0.182	4.62	
	¾″	5	0.220	0.530	13.5			4½	18	0.049	0.173	4.39	
	¾″	4	0.238	0.512	13 0			4½	17	0.058	0.164	4.17	
	⅝″	17	0.058	0.567	14.4			4½	16	0.065	0.157	3.99	
	⅝″	16	0.065	0.560	14.2			4½	15	0.072	0.150	3.81	r
	⅝″	15	0.072	0.553	14.1			4½	14	0.083	0.139	3.53	
	⅝″	14	0 083	0.542	13.8			4½	13	0.095	0.127	3.23	
	⅝″	13	0.095	0.530	13.5			4½	12	0.109	0.113	2 87	
	⅝″	12	0.109	0.516	13.1			5	24	0.025	0.175	4.44	
	⅝″	11	0.120	0.505	12.8	f		5	23	0.027	0.173	4.39	
	⅝″	10	0.134	0.491	12.5			5	22	0.0295	0.170	4 32	
	⅝″	9	0.148	0.477	12 1			5	21	0.0315	0.168	4.27	
	⅝″	8	0.165	0.460	11.7	r		5	20	0.035	0 165	4.19	
	⅝″	7	0.180	0.445	11.3	t		5	19	0.040	0.160	4.06	
	⅝″	6	0.203	0.422	10.7			5	18	0.049	0.151	3.84	
	⅝″	5	0.220	0.405	10.3			5	17	0.058	0.142	3.61	
Meshes to the inch.	2	18	0.049	0.451	11.46			5	16	0.065	0.135	3.43	
	2	17	0.058	0.442	11.23			5	15	0.072	0.128	3.25	
	2	16	0.065	0 435	11.05	d		5	14	0.083	0.117	2.97	
	2	15	0.072	0 428	10 87			5	13	0.095	0.105	2.67	
	2	14	0.083	0.417	10.59			6	25	0.023	0.143	3.63	
	2	13	0.095	0.405	10.29			6	24	0.025	0.141	3.58	
	2	12	0.109	0.391	9.93			6	23	0.027	0.139	3.53	
	2	11	0.120	0.380	9.65			6	22	0.0295	0.137	3.48	
	2	10	0.134	0.366	9 30			6	21	0.0315	0.135	3.43	
	2	9	0.148	0.352	8 94			6	20	0.035	0.131	3.33	
	2	8	0.165	0.335	8.51			6	19	0.040	0.126	3.20	f
	2½	19	0.040	0 360	9.14			6	18	0.049	0.117	2.97	
	2½	18	0.049	0.351	8.92			6	17	0.058	0.108	2.74	
	2½	17	0.058	0.342	8.69			6	16	0.065	0·101	2.57	r
	2½	16	0.065	0.335	8.51			6	15	0 072	0.094	2.39	
	2½	15	0.072	0.328	8.33			6	14	0.083	0.083	2.11	
	2½	14	0.083	0.317	8.05	t		7	26	0.0205	0.122	3.10	
	2½	13	0.095	0.305	7.75			7	25	0.023	0.120	3.05	
	2½	12	0.109	0.291	7.39	r		7	24	0.025	0.118	3.00	
	2½	11	0.120	0.280	7.11			7	23	0.027	0.116	2.95	
	2½	10	0.134	0.266	6.76			7	22	0.0295	0.113	2.87	
	2½	9	0.148	0.252	6.40			7	21	0.0315	0.111	2 82	t

d. De Kalb screens. *f.* Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *h* Richards (Hutchinson) screens. *p.* Nearest size to punched screens, Table II. *s.* Brunton's screens *t.* Sizes in Table I. *r.* Rittinger's screens.

TABLE IV.—*Continued.*

Mesh. Meshes to the inch.	Wire. Old English	Wire. Diameter in Inches	Space. Inches.	Space. Milli-meters.	Reference.
7	20	0.035	0 108	2.74	
7	19	0.040	0 103	2.62	
7	18	0.049	0.094	2.39	
7	17	0.058	0.085	2.16	
7	16	0.065	0.078	1.98	
7	15	0.072	0.071	1.80	
8	27	0.0187	0.106	2.69	
8	26	0.0205	0.104	2.64	
8	25	0.023	0.102	2.59	
8	24	0.025	0.100	2.54	
8	23	0.027	0.098	2.49	
8	22	0.0295	0.095	2.41	d
8	21	0.0315	0.093	2.36	
8	20	0.035	0.090	2.29	
8	19	0.040	0 085	2.16	
8	18	0.049	0 076	1.93	r
8	17	0.058	0.067	1.70	
8	16	0.065	0.060	1.52	
9	28	0.0165	0.095	2.40	
9	27	0.0187	0.092	2.35	
9	26	0.0205	0.091	2.30	
9	25	0.023	0.088	2.24	
9	24	0.025	0.086	2.19	
9	23	0.027	0.084	2.14	
9	22	0.0295	0 082	2.07	
9	21	0.0315	0.080	2.02	t
9	20	0.035	0.076	1.93	
9	19	0.040	0.071	1.81	
9	18	0.049	0.062	1.57	f
9	17	0.058	0 053	1.35	
10	29	0.0155	0 084	2.13	
10	28	0.0165	0.083	2.11	
10	27	0.0187	0.081	2 06	
10	26	0.0205	0.079	2.00	
10	25	0.023	0.077	1.96	
10	24	0.025	0.075	1.90	
10	23	0.027	0.073	1.85	d
10	22	0.0295	0.070	1.78	
10	21	0.0315	0.068	1.73	
10	20	0.035	0.065	1 65	
10	19	0.040	0.060	1.52	
10	18	0.049	0.051	1 29	
12	30	0.0137	0.070	1.78	
12	29	0.0155	0.068	1.73	
12	28	0.0165	0.067	1 70	
12	27	0.0187	0.065	1.65	
12	26	0.0205	0.063	1.60	
12	25	0.023	0.060	1.52	
12	24	0.025	0.058	1.47	pd
12	23	0.027	0.056	1.42	t
12	22	0.0295	0.054	1.37	r
12	21	0.0315	0.052	1.32	
12	20	0.035	0.048	1.22	
12	19	0.040	0.043	1.09	
14	34	0.0095	0.062	1.57	
14	33	0.0102	0.061	1.55	
14	32	0.0112	0.060	1.52	
14	31	0.0122	0.059	1.50	
14	30	0.0137	0.058	1.47	
14	29	0.0155	0.056	1.42	
14	28	0.0165	0.055	1.40	
14	27	0.0187	0.053	1.35	
14	26	0.0205	0.051	1.30	
14	25	0.0230	0.048	1.22	
14	24	0.0250	0.046	1.17	
14	23	0.0270	0.044	1.12	
14	22	0.0295	0.042	1.07	
14	21	0.0315	0.040	1.02	
14	20	0.0350	0.036	0.91	
16	35	0.0090	0.053	1.36	
16	34	0.0095	0.053	1.35	
16	33	0.0102	0.051	1.30	

Mesh. Meshes to the inch.	Wire. Old English.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
16	32	0.0112	0.050	1.27	
16	31	0.0122	0.049	1.25	
16	30	0.0137	0.049	1.24	p
16	29	0.0155	0.047	1.19	
16	28	0.0165	0.046	1.17	
16	27	0.0187	0.044	1.12	
16	26	0.0205	0.042	1.07	d
16	25	0.0230	0.039	0.99	rt
16	24	0.0250	0.037	0.94	
16	23	0.0270	0.035	0.89	
16	22	0.0295	0.033	0.84	
16	21	0.0315	0 031	0.79	
18	36	0.0075	0.048	1.22	
18	35	0.0090	0.046	1.18	
18	34	0.0095	0.046	1.17	
18	33	0.0102	0.045	1.14	
18	32	0.0112	0.044	1.12	
18	31	0.0122	0.043	1.09	
18	30	0.0137	0 042	1.07	p
18	29	0.0155	0.040	1.04	
18	28	0.0165	0.039	0.99	
18	27	0 0187	0.037	0.84	
18	26	0.0205	0.035	0.89	
18	25	0.0230	0.032	0.81	
18	24	0.0250	0.030	0.76	
18	23	0.0270	0.028	0.71	
18	22	0.0295	0.026	0.66	
20	36	0 0075	0.042	1.07	
20	35	0.0090	0.041	1.04	
20	34	0.0095	0.040	1.02	
20	33	0.0102	0.040	1.01	
20	32	0.0112	0.039	0.99	
20	31	0 0122	0.038	0.97	
20	30	0.0137	0.036	0.91	
20	29	0.0155	0.034	0.86	
20	28	0.0165	0.033	0.84	d
20	27	0.0187	0.031	0.79	f
20	26	0.0205	0.029	0.74	
20	25	0.0230	0.027	0.69	
20	24	0.0250	0.025	0.63	
20	23	0.0270	0.023	0.58	
20	22	0.0295	0.020	0.51	
20	21	0.0315	0.019	0.48	
20	20	0.0350	0.015	0.38	
22	36	0 0075	0.038	0.97	
22	35	0.0090	0.036	0.91	
22	34	0.0095	0.036	0 90	
22	33	0.0102	0.035	0.89	p
22	32	0 0112	0.034	0.86	
22	31	0.0122	0.033	0.84	
22	30	0.0137	0.031	0 79	
22	29	0.0155	0.030	0.76	
22	28	0.0165	0.029	0 74	
22	27	0.0187	0 026	0.66	
22	26	0 0205	0 025	0.64	
22	25	0.0230	0.022	0.56	
22	24	0.0250	0 020	0.51	
24	36	0.0075	0.034	0 86	
24	35	0.0090	0.033	0.83	
24	34	0.0095	0.032	0.81	
24	33	0.0102	0.030	0.76	
24	32	0.0112	0.029	0.74	p
24	31	0.0122	0.028	0.72	
24	30	0.0137	0.028	0.71	td
24	29	0.0155	0.026	0.66	
24	28	0.0165	0.025	0.63	r
24	27	0.0187	0.023	0.58	
24	26	0.0205	0.021	0.53	
24	25	0.0230	0.019	0.48	
26	36	0.0075	0.031	0.79	
26	35	0.0090	0.029	0.75	
26	34	0.0095	0.029	0.74	

d. De Kalb screens. *f.* Nearest approach to sizes of graded set, 1″′ ½″, ¼″, etc. *h.* Richards (Hutchinson) screens. *p.* Nearest size to punched screens, Table II. *s.* Brunton's screens. *t.* Sizes in Table I. *r.* Rittinger's screens.

Table IV.—*Concluded.*

Mesh. Meshes to the inch.	Wire. Old English.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
26	33	0.0102	0.028	0.71	
26	32	0.0112	0.027	0.69	p
26	31	0.0122	0.026	0.66	
26	30	0.0137	0.025	0.63	
26	29	0.0155	0.023	0.58	
26	28	0.0165	0.022	0.56	
26	27	0.0187	0.020	0.51	
26	26	0.0205	0.018	0.46	
28	36	0.0075	0.028	0.71	
28	35	0.0090	0.028	0.70	
28	34	0.0095	0.026	0.66	
28	33	0.0102	0.025	0.63	
28	32	0.0112	0.024	0.61	p
28	31	0.0122	0.023	0.58	
28	30	0.0137	0.022	0.56	
28	29	0.0155	0.020	0.51	
28	28	0.0165	0.019	0.48	
30	36	0.0075	0.026	0.65	
30	35	0.0090	0.024	0.61	
30	34	0.0095	0.024	0.60	
30	33	0.0102	0.023	0.58	
30	32	0.0112	0.022	0.56	p
30	31	0.0122	0.021	0.53	d
30	30	0.0137	0.020	0.50	tp
30	29	0.0155	0.018	0.46	
30	28	0.0165	0.017	0.43	
35	36	0.0075	0.021	0.53	
35	35	0.0090	0.019	0.50	
35	34	0.0095	0.019	0.48	
35	33	0.0102	0.018	0.46	p
35	32	0.0112	0.017	0.43	
35	31	0.0122	0.016	0.41	rp
35	30	0.0137	0.014	0.35	
35	29	0.0155	0.013	0.33	
40	36	0.0075	0.017	0.44	
40	35	0.0090	0.016	0.41	
40	34	0.0095	0.015	0.39	p
40	33	0.0102	0.015	0.37	fdh
40	32	0.0112	0.014	0.35	t
40	31	0.0122	0.013	0.32	
40	30	0.0137	0.011	0.28	
45	36	0.0075	0.015	0.37	
45	35	0.0090	0.013	0.33	
45	34	0.0095	0.013	0.32	
45	33	0.0102	0.012	0.30	
45	32	0.0112	0.011	0.28	
45	31	0.0122	0.010	0.25	
50	38	0.0057	0.014	0.36	
50	37	0.0065	0.013	0.34	p
50	36	0.0075	0.012	0.32	
50	35	0.0090	0.011	0.28	sd
50	34	0.0095	0.010	0.27	rh
55	38	0.0057	0.012	0.31	
55	37	0.0065	0.012	0.30	
55	36	0.0075	0.011	0.27	
55	35	0.0090	0.009	0.23	t
55	34	0.0095	0.009	0.22	
60	38	0.0057	0.011	0.28	
60	37	0.0065	0.010	0.26	
60	36	0.0075	0.009	0.23	d
60	35	0.0090	0.008	0.19	f
60	34	0.0095	0.007	0.18	s
64	39	0.0050	0.011	0.27	
64	38	0.0057	0.010	0.25	
64	37	0.0065	0.009	0.23	
64	36	0.0075	0.008	0.21	
64	35	0.0090	0.007	0.17	
70	40	0.0045	0.010	0.25	
70	39	0.0050	0.009	0.23	
70	38	0.0057	0.008	0.21	r
70	37	0.0065	0.008	0.20	d
70	36	0.0075	0.007	0.17	
74	41	0.0042	0.009	0.24	

Mesh. Meshes to the inch.	Wire. Old English.	Wire. Diameter in Inches.	Space. Inches	Space. Milli-meters	Reference.
74	40	0.0045	0.009	0.23	
74	39	0.0050	0.008	0.22	
74	38	0.0057	0.008	0.20	
74	37	0.0065	0.007	0.18	
80	41	0.0042	0.008	0.21	
80	40	0.0045	0.008	0.20	
80	39	0.0050	0.007	0.19	
80	38	0.0057	0.007	0.17	tsd
80	37	0.0065	0.0060	0.15	
90	41	0.0042	0.0068	0.17	
90	40	0.0045	0.0066	0.16	h
90	39	0.0050	0.0061	0.15	d
90	38	0.0057	0.0054	0.13	t
90	37	0.0065	0.0046	0.12	
100	43	0.0036	0.0064	0.163	
100	42	0.0040	0.0060	0.152	
100	41	0.0042	0.0057	0.146	
100	40	0.0045	0.0055	0.140	ds
100	39	0.0050	0.0050	0.127	
100	38	0.0057	0.0042	0.108	
110	42	0.0040	0.0051	0.153	
110	41	0.0042	0.0049	0.124	h
120	44	0.0032	0.0051	0.130	
120	43	0.0036	0.0047	0.120	
120	42	0.0040	0.0043	0.109	ds
130	44	0.0032	0.0045	0.114	
130	43	0.0036	0.0041	0.104	
140	45	0.0028	0.0043	0.110	
140	44	0.0032	0.0039	0.100	f
150	46	0.0024	0.0042	0.107	
150	45	0.0028	0.0038	0.097	
150	44½	0.0030	0.0036	0.091	d
150	44	0.0032	0.0034	0.086	t
150	43	0.0036	0.0030	0.076	
150	42	0.0040	0.0026	0.066	
160	45½	0.0026	0.0036	0.091	
160	45	0.0028	0.0034	0.086	
160	44½	0.0030	0.0032	0.081	
160	44	0.0032	0.0030	0.076	
160	43	0.0036	0.0026	0.066	t
160	42	0.0040	0.0022	0.056	
170	45½	0.0026	0.0033	0.084	
170	45	0.0028	0.0031	0.079	
170	44½	0.0030	0.0029	0.074	
170	44	0.0032	0.0027	0.069	
170	43	0.0036	0.0023	0.058	
170	42	0.0040	0.0019	0.048	
180	46½	0.0022	0.0033	0.084	
180	46	0.0024	0.0031	0.079	
180	45½	0.0026	0.0029	0.074	h
180	45	0.0028	0.0027	0.069	
180	44½	0.0030	0.0025	0.063	
180	44	0.0032	0.0023	0.058	
180	43	0.0036	0.0019	0.048	t
180	42	0.0040	0.0015	0.038	
190	47	0.0020	0.0032	0.081	
190	46½	0.0022	0.0030	0.076	
190	46	0.0024	0.0028	0.071	
190	45½	0.0026	0.0026	0.066	
190	45	0.0028	0.0024	0.061	
190	44½	0.0030	0.0022	0.056	
190	44	0.0032	0.0020	0.051	
190	43	0.0036	0.0016	0.041	
190	42	0.0040	0.0012	0.030	t
200	47	0.0020	0.0030	0.076	
200	46½	0.0022	0.0028	0.071	
200	46	0.0024	0.0026	0.066	
200	45½	0.0026	0.0024	0.061	
200	45	0.0028	0.0022	0.056	
200	44½	0.0030	0.0020	0.051	f
200	44	0.0032	0.0018	0.046	
200	43	0.0036	0.0014	0.036	
200	42	0.0040	0.0010	0.025	t

d. De Kalb screens. *f*. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *h*. Richards (Hutchinson) screens *p*. Nearest size to punched screens, Table II. *s*. Brunton's screens. *t*. Sizes in Table I. *r*. Rittinger's screens.

TABLE V.—*Sizes of Wire Screens (mostly of brass in the coarse sizes) of the Birmingham (or Stub's) Gauge.*

Mesh.	Wire. B. or S.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
Inches to the mesh.					
3″	0	0.340	2.66	67.6	
3″	00	0 380	2.62	66.5	r
3″	000	0 425	2.57	65.3	t
2½″	1	0.300	2.20	55.9	
2½″	0	0.340	2.16	54.9	
2½″	00	0.380	2.12	53.8	f
2⅛″	0	0.340	1.78	45 2	rt
1½″	3	0.259	1.24	31.5	t
1½″	2	0.284	1.22	31.0	r
1½″	1	0.300	1.20	30.5	
1⅛″	4	0.238	0.887	22.5	rt
1″	15	0.072	0.928	23 6	f
1″	14	0.083	0 917	23.3	
1″	13	0.095	0.905	23.0	
1″	12	0.109	0.891	22.6	
1″	11	0.120	0.880	22.3	
1″	10	0.134	0.866	22 0	
1″	9	0.148	0.852	21.6	
1″	8	0.165	0.835	21.2	
1″	7	0.180	0.820	20.8	
1″	6	0.203	0.797	20.2	
1″	5	0.220	0.780	19.8	
1″	4	0.238	0.762	19.3	
1″	3	0.259	0.741	18.8	
¾″	16	0.065	0.685	17.4	
¾″	15	0 072	0.678	17.2	
¾″	14	0.083	0.667	16.9	
¾″	13	0.095	0.655	16.6	
¾″	12	0.109	0.641	16 3	
¾″	11	0.120	0.630	16 0	t
¾″	10	0.134	0.616	15.6	
¾″	9	0.148	0 602	15.3	
¾″	8	0.165	0.585	14 9	
¾″	7	0.180	0.570	14.5	
¾″	6	0.203	0.547	13.9	
¾″	5	0.220	0.530	13.5	
¾″	4	0 238	0.512	13 0	
⅝″	17	0.058	0.567	14.4	
⅝″	16	0.065	0.560	14.2	
⅝″	15	0.072	0.553	14.0	
⅝″	14	0.083	0.542	13.8	
⅝″	13	0.095	0.530	13.5	
⅝″	12	0.109	0.516	13.1	
⅝″	11	0.120	0.505	12.8	f
⅝″	10	0.134	0.491	12 5	
⅝″	9	0.148	0.477	12.1	
⅝″	8	0.165	0.460	11.7	r
⅝″	7	0.180	0.445	11.3	t
⅝″	6	0.203	0.422	10 7	
⅝″	5	0.220	0.405	10.3	
Meshes to the inch.					
2	18	0.049	0.451	11 5	
2	17	0.058	0.442	11.2	
2	16	0.065	0.435	11 0	
2	15	0.072	0.428	10 9	
2	14	0.083	0.417	10.6	
2	13	0.095	0.405	10 3	
2	12	0.109	0.391	9.93	
2	11	0.120	0.380	9.65	
2	10	0.134	0.366	9.30	
2	9	0.148	0.352	8.94	
2	8	0.165	0.335	8.51	
2½	19	0.042	0.358	9.09	
2½	18	0.049	0.351	8.92	
2½	17	0.058	0.342	8.69	
2½	16	0.065	0.335	8.51	
2½	15	0.072	0.328	8.33	
2½	14	0.083	0.317	8.05	t
2½	13	0.095	0.305	7.75	
2½	12	0.109	0.291	7.39	r
2½	11	0.120	0.280	7.11	
2½	10	0.134	0 266	6.76	
2½	9	0.148	0.252	6.40	

Mesh.	Wire. B. or S.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
Meshes to the inch.					
3	20	0.035	0.298	7.57	
3	19	0 042	0.291	7.39	
3	18	0.049	0.284	7.21	
3	17	0.058	0 275	6 98	
3	16	0.065	0 268	6 81	
3	15	0.072	0.261	6 63	
3	14	0.083	0 250	6.35	
3	13	0.095	0.238	6.05	
3	12	0 109	0 224	5.69	
3	11	0.120	0.213	5.41	
3	10	0.134	0 199	5.05	
3½	20	0.035	0.250	6.35	
3½	19	0.042	0 243	6.17	
3½	18	0.049	0 236	5.99	
3½	17	0.058	0.227	5.77	
3½	16	0.065	0 220	5 59	t
3½	15	0.072	0.213	5 41	
3½	14	0.083	0 202	5.13	r
3½	13	0.095	0 190	4.83	
3½	12	0.109	0.176	4.48	
3½	11	0.120	0.165	4.19	
4	21	0.032	0 218	5.54	
4	20	0.035	0.215	5.46	
4	19	0.042	0.208	5.28	
4	18	0.049	0.201	5.11	
4	17	0.058	0 192	4.88	
4	16	0.065	0.185	4.70	
4	15	0.072	0.178	4.52	
4	14	0.083	0.167	4.24	
4	13	0.095	0.155	3.94	
4	12	0.109	0.141	3 58	
4	11	0.120	0 130	3.29	
4½	21	0.032	0.190	4.83	
4½	20	0.035	0.187	4.75	
4½	19	0.042	0.180	4.57	
4½	18	0.049	0.173	4.39	
4½	17	0.058	0.164	4.17	
4½	16	0.065	0.157	3.99	t
4½	15	0.072	0.150	3.81	r
4½	14	0.083	0.139	3.53	
4½	13	0.095	0.127	3.23	
4½	12	0.109	0.113	2.84	
5	21	0.022	0.178	4.52	
5	23	0.025	0.175	4.44	
5	22	0.028	0.172	4.37	
5	21	0.032	0 168	4.27	
5	20	0.035	0.165	4.19	
5	19	0.042	0.158	4.01	
5	18	0.049	0.151	3.84	
5	17	0.058	0.142	3 61	
5	16	0.065	0.135	3.43	
5	15	0.072	0.128	3.24	
5	14	0.083	0.117	2.97	
5	13	0.095	0.105	2.67	
6	25	0.020	0.146	3.71	
6	24	0.022	0.144	3.66	
6	23	0.025	0.141	3.58	
6	22	0.028	0.138	3.51	
6	21	0.032	0.134	3.40	
6	20	0.035	0.131	3.33	
6	19	0.042	0.124	3.15	f
6	18	0.049	0.117	2.97	
6	17	0.058	0.108	2.74	
6	16	0.065	0.101	2.57	r
6	15	0.072	0.094	2.39	
6	14	0.083	0.083	2.11	
7	26	0.018	0.125	3.17	
7	25	0 020	0.123	3.12	
7	24	0.022	0.121	3.07	
7	23	0.025	0.118	3.00	
7	22	0.028	0.115	2.92	
7	21	0.032	0.111	2.82	t

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *p.* Nearest size to punched screens, Table II. *t.* Sizes in Table I. *r.* Rittinger's screens.

TABLE V.—*Continued.*

Mesh (Meshes to the inch.)	Wire. B. or S.	Wire. Diameter in Inches.	Space. Inches.	Space. Millimeters.	Reference.
7	20	0.035	0.108	2.74	
7	19	0.042	0.101	2.57	
7	18	0.049	0.094	2.32	
7	17	0.058	0.085	2.16	
7	16	0.065	0.078	1.98	
7	15	0.072	0.071	1.80	
8	27	0.016	0.109	2.77	
8	26	0.018	0.107	2.72	
8	25	0.020	0.105	2.67	
8	24	0.022	0.103	2.62	
8	23	0.025	0.100	2.54	
8	22	0.028	0.097	2.46	
8	21	0.032	0.093	2.36	
8	20	0.035	0.090	2.29	
8	19	0.042	0.083	2.11	
8	18	0.049	0.076	1.93	r
8	17	0.058	0.067	1.70	
8	16	0.065	0.060	1.52	
9	28	0.014	0.097	2.46	
9	27	0.016	0.095	2.41	
9	26	0.018	0.093	2.36	
9	25	0.020	0.091	2.31	
9	24	0.022	0.089	2.27	
9	23	0.025	0.086	2.18	
9	22	0.028	0.083	2.11	
9	21	0.032	0.079	2.01	t
9	20	0.035	0.076	1.93	
9	19	0.042	0.069	1.75	
9	18	0.049	0.062	1.57	
9	17	0.058	0.053	1.35	
10	29	0.013	0.087	2.21	
10	28	0.014	0.086	2.18	
10	27	0.016	0.084	2.13	
10	26	0.018	0.082	2.08	
10	25	0.020	0.080	2.03	
10	24	0.022	0.078	1.98	
10	23	0.025	0.075	1.90	
10	22	0.028	0.072	1.83	
10	21	0.032	0.068	1.73	
10	20	0.035	0.065	1.65	f
10	19	0.042	0.058	1.47	
10	18	0.049	0.051	1.30	
12	30	0.012	0.071	1.80	
12	29	0.013	0.070	1.78	
12	28	0.014	0.069	1.75	
12	27	0.016	0.067	1.70	
12	26	0.018	0.065	1.65	
12	25	0.020	0.063	1.60	
12	24	0.022	0.061	1.55	
12	23	0.025	0.058	1.47	p
12	22	0.028	0.055	1.40	rt
12	21	0.032	0.051	1.30	
12	20	0.035	0.048	1.22	
12	19	0.042	0.041	1.04	
14	30	0.012	0.059	1.50	
14	29	0.013	0.058	1.47	
14	28	0.014	0.057	1.45	
14	27	0.016	0.055	1.40	
14	26	0.018	0.053	1.35	
14	25	0.020	0.051	1.30	
14	24	0.022	0.049	1.24	p
14	23	0.025	0.046	1.17	
14	22	0.028	0.044	1.12	
14	21	0.032	0.039	0.99	
14	20	0.036	0.036	0.91	
16	30	0.012	0.050	1.27	
16	29	0.013	0.049	1.24	
16	28	0.014	0.048	1.22	
16	27	0.016	0.046	1.17	
16	26	0.018	0.044	1.12	
16	25	0.020	0.042	1.07	rp
16	24	0.022	0.040	1.02	t

Mesh. (Meshes to the inch.)	Wire. B. or S.	Wire. Diameter in Inches.	Space. Inches.	Space. Millimeters.	Reference.
16	23	0.025	0.037	0.94	
16	22	0.028	0.034	0.86	
16	21	0.032	0.030	0.76	
16	20	0.035	0.027	0.69	
18	31	0.010	0.045	1.14	
18	30	0.012	0.043	1.09	
18	29	0.013	0.042	1.07	
18	28	0.014	0.041	1.04	
18	27	0.016	0.039	0.99	
18	26	0.018	0.037	0.94	
18	25	0.020	0.035	0.89	
18	24	0.022	0.033	0.84	
18	23	0.025	0.030	0.76	
18	22	0.028	0.027	0.69	
18	21	0.032	0.023	0.58	
20	32	0.009	0.041	1.04	
20	31	0.010	0.040	1.01	
20	30	0.012	0.038	0.96	
20	29	0.013	0.037	0.94	
20	28	0.014	0.036	0.91	
20	27	0.016	0.034	0.86	
20	26	0.018	0.032	0.81	
20	25	0.020	0.030	0.76	
20	24	0.022	0.028	0.71	
20	23	0.025	0.025	0.63	
20	22	0.028	0.022	0.56	
22	32	0.009	0.036	0.91	
22	31	0.010	0.035	0.89	p
22	30	0.012	0.033	0.84	
22	29	0.013	0.032	0.81	
22	28	0.014	0.031	0.79	f
22	27	0.016	0.029	0.74	
22	26	0.018	0.027	0.69	
22	25	0.020	0.025	0.64	
22	24	0.022	0.023	0.58	
22	23	0.025	0.020	0.51	
24	34	0.007	0.035	0.89	
24	33	0.008	0.034	0.86	
24	32	0.009	0.033	0.84	
24	31	0.010	0.032	0.81	
24	30	0.012	0.030	0.76	
24	29	0.013	0.029	0.74	
24	28	0.014	0.028	0.71	rt
24	27	0.016	0.026	0.66	
24	26	0.018	0.023	0.61	
24	25	0.020	0.022	0.56	
24	24	0.022	0.020	0.51	
24	23	0.025	0.017	0.43	
26	34	0.007	0.031	0.79	
26	33	0.008	0.030	0.76	
26	32	0.009	0.029	0.74	p
26	31	0.010	0.028	0.71	
26	30	0.012	0.026	0.66	
26	29	0.013	0.025	0.63	
26	28	0.014	0.024	0.61	
26	27	0.016	0.022	0.56	
26	26	0.018	0.020	0.51	
26	25	0.020	0.018	0.46	
26	24	0.022	0.016	0.41	
28	34	0.007	0.029	0.73	
28	33	0.008	0.028	0.71	
28	32	0.009	0.027	0.68	p
28	31	0.010	0.026	0.66	
28	30	0.012	0.024	0.61	
28	29	0.013	0.023	0.58	
28	28	0.014	0.022	0.56	
28	27	0.016	0.020	0.51	
28	26	0.018	0.018	0.46	
28	25	0.020	0.016	0.41	
30	34	0.007	0.026	0.66	
30	33	0.008	0.025	0.63	
30	32	0.009	0.024	0.61	p

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *p.* Nearest size to punched screens, Table II. *t.* Sizes in Table I. *r.* Rittinger's screens.

Table V.—*Concluded.*

Mesh. (Meshes to the inch.)	Wire. B. or S.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
30	31	0.010	0.023	0.58	p
30	30	0.012	0.021	0.53	
30	29	0.013	0.020	0.51	tp
30	28	0.014	0.019	0.48	
30	27	0.016	0.017	0.43	
30	26	0.018	0.015	0.38	
30	25	0.020	0.013	0.33	
35	34	0.007	0.021	0.53	
35	33	0.008	0.020	0.51	
35	32	0.009	0.019	0.48	
35	31	0.010	0.018	0.46	rp
35	30	0.012	0.016	0.41	p
35	29	0.013	0.015	0.38	f
35	28	0.014	0.014	0.35	
35	27	0.016	0.012	0.30	
35	26	0.018	0.010	0.25	
40	34	0.007	0.018	0.46	
40	33	0.008	0.017	0.43	
40	32	0.009	0.016	0.41	
40	31	0.010	0.015	0.38	p
40	30	0.012	0.013	0.33	t
40	29	0.013	0.012	0.30	
40	28	0.014	0.011	0.28	
45	34	0.007	0.015	0.38	
45	33	0.008	0.014	0.35	p
45	32	0.009	0.013	0.33	
45	31	0.010	0.012	0.30	
45	30	0.012	0.010	0.25	
45	29	0.013	0.009	0.23	
50	34	0.007	0.013	0.33	r
50	33	0.008	0.012	0.30	
50	32	0.009	0.011	0.28	
50	31	0.010	0.010	0.25	t
50	30	0.012	0.008	0.20	
55	34	0.007	0.011	0.28	
55	33	0.008	0.010	0.25	
55	32	0.009	0.009	0.23	

Mesh. (Meshes to the inch.)	Wire. B. or S.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
55	31	0.010	0.008	0.20	
55	30	0.012	0.006	0.15	
60	34	0.007	0.010	0.25	
60	33	0.008	0.009	0.23	
60	32	0.009	0.008	0.20	f
60	31	0.010	0.007	0.18	
60	30	0.012	0.005	0.13	
64	34	0.007	0.009	0.23	
64	33	0.008	0.008	0.20	
64	32	0.009	0.007	0.18	t
64	31	0.010	0.006	0.15	
64	30	0.012	0.004	0.10	
70	35	0.005	0.009	0.23	
70	34	0.007	0.007	0.18	
70	33	0.008	0.006	0.15	
70	32	0.009	0.005	0.13	
70	31	0.010	0.004	0.10	
74	35	0.005	0.008	0.20	
74	34	0.007	0.006	0.15	
74	33	0.008	0.005	0.13	t
74	32	0.009	0.004	0.10	
74	31	0.010	0.003	0.08	
80	36	0.004	0.008	0.20	
80	35	0.005	0.007	0.18	
80	34	0.007	0.005	0.13	
80	33	0.008	0.004	0.10	
80	32	0.009	0.003	0.08	
90	36	0.004	0.007	0.18	
90	35	0.005	0.006	0.15	
90	34	0.007	0.004	0.10	f
90	33	0.008	0.003	0.08	
90	32	0.009	0.002	0.05	
100	36	0.004	0.006	0.15	
100	35	0.005	0.005	0.13	
100	34	0.007	0.003	0.08	t
100	33	0.008	0.002	0.05	

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *p*. Nearest size to punched screens, Table II. *t*. Sizes in Table I. *r*. Rittinger's screens.

TABLE VI.—*Sizes of Wire Screens (usually copper) of the American (or Brown & Sharpe) Gauge.*

Inches to the mesh.

Mesh.	Wire. B. & S	Wire. Diameter in Inches	Space. Inches.	Space. Milli-meters.	Reference.
3″	0	0.3248	2.67	68.0	
3″	00	0.3648	2.63	66.8	r
3″	000	0.4096	2.59	65.8	t
2½″	1	0.2893	2.21	56.1	
2½″	0	0.3248	2.17	55.2	
2½″	00	0.3648	2.13	54.2	f
2⅛″	0	0.3248	1.80	45.7	rt
1½″	3	0.2294	1.27	32.3	t
1½″	2	0.2576	1.24	31.5	r
1½″	1	0.2893	1.21	30.7	
1⅛″	4	0.2043	0.92	23.4	rt
1″	15	0.0570	0.943	23.9	f
1″	14	0.0640	0.936	23.8	
1″	13	0 0719	0.928	23.6	
1″	12	0.0808	0.919	23.3	
1″	11	0.0907	0.909	23.1	
1″	10	0.1018	0.898	22.8	
1″	9	0.1144	0.886	22.5	
1″	8	0.1284	0.872	22.1	
1″	7	0.1442	0.856	21.7	
1″	6	0.1620	0.838	21.3	
1″	5	0.1819	0.818	20 8	
1″	4	0.2043	0.796	20.2	
1″	3	0.2294	0.771	19.6	
¾″	16	0.0508	0.699	17.7	
¾″	15	0.0570	0.693	17.6	
¾″	14	0 0641	0.686	17.4	
¾″	13	0.0719	0.678	17.2	
¾″	12	0.0808	0.669	17.0	
¾″	11	0.0907	0.659	16.7	
¾″	10	0.1018	0.648	16 5	
¾″	9	0.1144	0.636	16.1	t
¾″	8	0.1284	0.622	15.8	
¾″	7	0.1442	0.606	15.4	
¾″	6	0.1620	0.588	14.9	
¾″	5	0.1819	0.568	14 4	
¾″	4	0.2043	0.546	13.9	
⅝″	17	0.0452	0.580	14.7	
⅝″	16	0.0508	0.574	14.5	
⅝″	15	0.0570	0.568	14.4	
⅝″	14	0 0640	0.561	14.2	
⅝″	13	0.0719	0.553	14.0	
⅝″	12	0.0808	0.544	13 8	
⅝″	11	0.0907	0.534	13.6	
⅝″	10	0.1018	0 523	13.3	
⅝″	9	0.1144	0.510	12.9	
⅝″	8	0.1284	0.497	12.6	rf
⅝″	7	0.1442	0.481	12.2	
⅝″	6	0.1620	0.463	11.8	
⅝″	5	0.1819	0.443	11.2	

Meshes to the inch.

Mesh.	Wire. B. & S	Wire. Diameter in Inches	Space. Inches.	Space. Milli-meters.	Reference.
2	19	0.0358	0.464	11.79	
2	18	0.0403	0.460	11.68	
2	17	0.0452	0.455	11.56	
2	16	0.0508	0.449	11.40	
2	15	0.0570	0.443	11.25	t
2	14	0.0640	0.436	11.07	
2	13	0.0719	0 428	10.87	
2	12	0.0808	0.419	10.64	
2	11	0.0907	0.409	10.39	
2	10	0.1018	0.398	10.11	
2	9	0.1144	0.386	9.80	
2	8	0.1284	0.372	9.45	
2	7	0.1442	0.356	9.04	
2	6	0.1620	0.338	8.59	
2½	19	0.0358	0.364	9.25	
2½	18	0.0403	0.360	9.14	
2½	17	0.0452	0.355	9.02	
2½	16	0.0508	0.349	8.86	
2½	15	0.0570	0.343	8.71	
2½	14	0.0640	0.336	8.53	
2½	13	0.0719	0.328	8.33	
2½	12	0.0808	0.319	8.10	rt

Mesh.	Wire. B. & S.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
2½	11	0.0907	0.309	7.85	
2½	10	0.1018	0.298	7.57	
2½	9	0.1144	0.286	7.26	
2½	8	0.1284	0.272	6.91	
3	20	0.0319	0.301	7.65	
3	19	0.0358	0.297	7.54	
3	18	0.0403	0.293	7.44	
3	17	0.0452	0.288	7.32	
3	16	0.0508	0.282	7.16	
3	15	0.0570	0.276	7.01	
3	14	0.0640	0.269	6.83	
3	13	0.0719	0.261	6.63	
3	12	0.0808	0.252	6.40	f
3	11	0.0907	0.243	6.17	
3	10	0.1018	0.231	5.87	
3	9	0.1144	0.220	5.59	
3	8	0.1284	0.205	5.21	
3½	21	0.0284	0.257	6.53	
3½	20	0.0319	0.254	6.45	
3½	19	0.0358	0.250	6.35	
3½	18	0.0403	0.244	6.20	
3½	17	0.0452	0.239	6.07	
3½	16	0.0508	0.235	5.97	
3½	15	0.0570	0.229	5.82	
3½	14	0.0640	0.222	5.64	rt
3½	13	0.0719	0.213	5.41	
3½	12	0 0808	0.205	5.20	
3½	11	0.0907	0.195	4.95	
4	22	0.0253	0.225	5.71	
4	21	0.0284	0.222	5.64	
4	20	0.0319	0.218	5.54	
4	19	0.0358	0.214	5.44	
4	18	0.0403	0.210	5.33	
4	17	0.0452	0.207	5.26	
4	16	0.0508	0.199	5.05	
4	15	0.0570	0.193	4.90	
4	14	0.0640	0.186	4.72	
4	13	0.0719	0.178	4.52	
4	12	0.0808	0.169	4.29	
4	11	0.0907	0.159	4.04	
4	10	0.1018	0.148	3.76	
4	9	0.1144	0.136	3.45	
4½	23	0.0225	0.200	5.08	
4½	22	0.0253	0.197	5.00	
4½	21	0.0284	0.194	4.93	
4½	20	0.0319	0.190	4.83	
4½	19	0.0358	0.186	4.72	
4½	18	0.0403	0.182	4.62	
4½	17	0.0452	0.177	4.50	
4½	16	0.0508	0.171	4.34	
4½	15	0.0570	0.165	4.19	rt
4½	14	0.0640	0.158	4.01	
4½	13	0.0719	0.150	3.81	
4½	12	0.0808	0.141	3.58	
5	23	0.0225	0.177	4.50	
5	22	0.0253	0.175	4.44	
5	21	0.0284	0.172	4.37	
5	20	0.0319	0.168	4.27	
5	19	0.0358	0.164	4.17	
5	18	0.0403	0.160	4.06	
5	17	0.0452	0.155	3.94	
5	16	0.0508	0.149	3.78	
5	15	0.0570	0.143	3.63	
5	14	0.0648	0.135	3.43	
5	13	0.0719	0.128	3.25	
5	12	0.0808	0.119	3.02	
5	11	0.0907	0.109	2.77	
6	25	0.0179	0.149	3.78	
6	24	0.0201	0.146	3.71	
6	23	0.0225	0.144	3.66	
6	22	0.0253	0.141	3.58	
6	21	0.0284	0.138	3.50	

f. Nearest approacn to sizes of graded set, 1″, ½″, ¼″, etc. *p.* Nearest size to punched screens, Table II. *t.* Sizes in Table I. *r.* Rittinger's screens.

TABLE VI.—*Continued.*

Mesh. (Meshes to the inch.)	Wire. B. & S.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.	Mesh. (Meshes to the inch.)	Wire. B. & S.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
6	20	0.0319	0.135	3.43		12	19	0.0358	0.047	1.19	
6	19	0.0358	0.131	3.33		12	18	0.0403	0.043	1.09	
6	18	0.0403	0.126	3.20	f	14	34	0.0063	0.065	1.65	
6	17	0.0452	0.121	3.07		14	33	0.0070	0.064	1.63	
6	16	0.0508	0.116	2.95	r	14	32	0.0079	0.063	1.60	
6	15	0.0570	0.110	2.79		14	31	0.0089	0.062	1.57	
6	14	0.0640	0.102	2.59		14	30	0.0100	0.061	1.55	
6	13	0.0719	0.095	2.41		14	29	0.0112	0.060	1.52	
6	12	0.0808	0.086	2.18		14	28	0.0126	0.059	1.50	
7	26	0.0159	0.127	3.23		14	27	0.0141	0.057	1.45	
7	25	0.0179	0.125	3.17		14	26	0.0159	0.055	1.40	
7	24	0.0201	0.123	3.12		14	25	0.0179	0.053	1.35	
7	23	0.0225	0.120	3.05		14	24	0.0201	0.051	1.30	
7	22	0.0253	0.117	2.97		14	23	0.0225	0.049	1.24	
7	21	0.0284	0.114	2.90		14	22	0.0253	0.046	1.17	
7	20	0.0319	0.111	2.82	t	14	21	0.0284	0.043	1.09	
7	19	0.0358	0.107	2.72		14	20	0.0319	0.039	0.99	
7	18	0.0403	0.102	2.59		16	35	0.0056	0.057	1.45	
7	17	0.0452	0.098	2.49		16	34	0.0063	0.056	1.42	
7	16	0.0508	0.092	2.34		16	33	0.0070	0.055	1.40	
7	15	0.0570	0.086	2.18		16	32	0.0079	0.054	1.37	
8	27	0.0141	0.111	2.82		16	31	0.0089	0.053	1.35	
8	26	0.0159	0.109	2.77		16	30	0.0100	0.052	1.32	
8	25	0.0179	0.107	2.72		16	29	0.0112	0.051	1.30	
8	24	0.0201	0.105	2.67		16	28	0.0126	0.050	1.27	
8	23	0.0225	0.102	2.59		16	27	0.0141	0.048	1.23	p
8	22	0.0253	0.100	2.54		16	26	0.0159	0.047	1.19	
8	21	0.0284	0.097	2.46		16	25	0.0179	0.045	1.14	r
8	20	0.0319	0.093	2.36		16	24	0.0201	0.042	1.07	
8	19	0.0358	0.089	2.26		16	23	0.0225	0.040	1.02	t
8	18	0.0403	0.085	2.16	r	16	22	0.0253	0.037	0.94	
8	17	0.0452	0.080	2.03		16	21	0.0284	0.034	0.86	
8	16	0.0508	0.074	1.88		18	36	0.0050	0.051	1.28	
8	15	0.0570	0.068	1.73		18	35	0.0056	0.050	1.27	
8	14	0.0640	0.061	1.55		18	34	0.0063	0.049	1.25	
9	28	0.0126	0.098	2.49		18	33	0.0070	0.048	1.23	
9	27	0.0141	0.096	2.44		18	32	0.0079	0.047	1.21	
9	26	0.0159	0.095	2.41		18	31	0.0089	0.046	1.18	
9	25	0.0179	0.093	2.36		18	30	0.0100	0.045	1.17	
9	24	0.0201	0.091	2.31		18	29	0.0112	0.044	1.13	
9	23	0.0225	0.089	2.26		18	28	0.0126	0.042	1.09	
9	22	0.0253	0.086	2.18		18	27	0.0141	0.041	1.04	
9	21	0.0284	0.083	2.11		18	26	0.0159	0.039	0.98	
9	20	0.0319	0.079	2.00	t	18	25	0.0179	0.038	0.97	
9	19	0.0358	0.075	1.90		18	24	0.0201	0.035	0.89	
9	18	0.0403	0.071	1.80		18	23	0.0225	0.033	0.84	
9	17	0.0452	0.066	1.68		18	22	0.0253	0.030	0.76	
10	29	0.0112	0.089	2.26		20	36	0.0050	0.045	1.14	
10	28	0.0126	0.087	2.21		20	35	0.0056	0.044	1.13	
10	27	0.0141	0.086	2.18		20	34	0.0063	0.043	1.12	
10	26	0.0159	0.084	2.13		20	33	0.0070	0.042	1.09	
10	25	0.0179	0.082	2.08		20	32	0.0079	0.042	1.07	p
10	24	0.0201	0.080	2.03		20	31	0.0089	0.041	1.04	
10	23	0.0225	0.077	1.96		20	30	0.0100	0.039	1.01	
10	22	0.0253	0.075	1.90		20	29	0.0112	0.038	0.99	
10	21	0.0284	0.072	1.83		20	28	0.0126	0.037	0.95	
10	20	0.0319	0.068	1.73		20	27	0.0141	0.035	0.91	
10	19	0.0358	0.064	1.63	f	20	26	0.0159	0.034	0.86	
10	18	0.0403	0.060	1.52		20	25	0.0179	0.032	0.82	
10	17	0.0452	0.055	1.40		20	24	0.0201	0.029	0.76	
10	16	0.0508	0.049	1.24		20	23	0.0225	0.027	0.70	
12	30	0.0100	0.073	1.85		22	36	0.0050	0.040	1.02	
12	29	0.0112	0.072	1.84		22	35	0.0056	0.039	0.99	
12	28	0.0126	0.071	1.83		22	34	0.0063	0.039	0.98	
12	27	0.0141	0.069	1.75		22	33	0.0070	0.038	0.97	
12	26	0.0159	0.067	1.70		22	32	0.0079	0.037	0.94	
12	25	0.0179	0.065	1.65		22	31	0.0089	0.036	0.91	
12	24	0.0201	0.063	1.60		22	30	0.0100	0.035	0.89	
12	23	0.0225	0.061	1.55		22	29	0.0112	0.034	0.86	
12	22	0.0254	0.058	1.47	rp	22	28	0.0126	0.032	0.81	
12	21	0.0284	0.055	1.40	t	22	27	0.0141	0.031	0.79	
12	20	0.0319	0.051	1.30		22	26	0.0159	0.029	0.74	

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *p.* Nearest size to punched screens, Table II. *t.* Sizes in Table I. *r.* Rittinger's screens.

Table VI.—*Concluded.*

Mesh. Meshes to the inch.	Wire. B. & S.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.	Mesh. Meshes to the inch.	Wire. B. & S.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
22	25	0.0179	0.027	0.69		40	37	0.0044	0.021	0.52	
22	24	0.0201	0.025	0.64		40	36	0.0050	0.020	0.51	
24	36	0.0050	0.036	0.93		40	35	0.0056	0.019	0.49	
24	35	0.0056	0.036	0.91		40	34	0.0063	0.019	0.47	
24	34	0.0063	0.035	0.90	p	40	33	0.0070	0.018	0.46	p
24	33	0.0070	0.034	0.88		40	32	0.0079	0.017	0.43	p
24	32	0.0079	0.033	0.86		40	31	0.0089	0.016	0.41	f
24	31	0.0089	0.032	0.83		40	30	0.0100	0.015	0.38	
24	30	0.0100	0.031	0.80		45	37	0.0044	0.018	0.45	
24	29	0.0112	0.030	0.77		45	36	0.0050	0.017	0.44	
24	28	0.0126	0.029	0.74	r	45	35	0.0056	0.017	0.42	
24	27	0.0141	0.027	0.70	t	45	34	0.0063	0.016	0.40	
24	26	0.0159	0.025	0.65		45	33	0.0070	0.015	0.39	
24	25	0.0179	0.023	0.60		45	32	0.0079	0.014	0.36	
26	36	0.0050	0.034	0.85		50	37	0.0044	0.016	0.40	
26	35	0.0056	0.033	0.84		50	36	0.0050	0.015	0.38	p
26	34	0.0063	0.032	0.82		50	35	0.0056	0.014	0.37	
26	33	0.0070	0.031	0.80	t	50	34	0.0063	0.014	0.35	rtp
26	32	0.0079	0.030	0.77		50	33	0.0070	0.013	0.33	
26	31	0.0089	0.029	0.75	p	50	32	0.0079	0.012	0.31	
26	30	0.0100	0.028	0.72		60	38	0.0039	0.013	0.32	
26	29	0.0112	0.027	0.69	p	60	37	0.0044	0.012	0.31	
26	28	0.0126	0.025	0.66		60	36	0.0050	0.012	0.29	
26	27	0.0141	0.024	0.61		60	35	0.0056	0.011	0.28	
26	26	0.0159	0.022	0.57		60	34	0.0063	0.010	0.26	
28	36	0.0050	0.031	0.78		64	38	0.0039	0.012	0.30	
28	35	0.0056	0.030	0.76		64	37	0.0044	0.011	0.28	
28	34	0.0063	0.029	0.75		64	36	0.0050	0.011	0.27	
28	33	0.0070	0.029	0.73		64	35	0.0056	0.010	0.25	t
28	32	0.0079	0.028	0.71		64	34	0.0063	0.009	0.24	
28	31	0.0089	0.027	0.68		70	38	0.0039	0.010	0.26	r
28	30	0.0100	0.025	0.65		70	37	0.0044	0.010	0.25	
28	29	0.0112	0.024	0.62	p	70	36	0.0050	0.009	0.23	
28	28	0.0126	0.023	0.59		70	35	0.0056	0.009	0.22	
28	27	0.0141	0.021	0.55		70	34	0.0063	0.008	0.20	
30	36	0.0050	0.028	0.72		74	38	0.0039	0.010	0.24	
30	35	0.0056	0.028	0.70		74	37	0.0044	0.009	0.23	
30	34	0.0063	0.027	0.69		74	36	0.0050	0.008	0.22	
30	33	0.0070	0.026	0.67		74	35	0.0056	0.008	0.20	f
30	32	0.0079	0.025	0.65		74	34	0.0063	0.007	0.18	
30	31	0.0089	0.024	0.62		80	39	0.0035	0.009	0.23	
30	30	0.0100	0.023	0.59		80	38	0.0039	0.009	0.22	
30	29	0.0112	0.022	0.56	p	80	37	0.0044	0.008	0.21	
30	28	0.0126	0.021	0.53		80	36	0.0050	0.007	0.19	
30	27	0.0141	0.019	0.49		80	35	0.0056	0.007	0.18	
35	37	0.0044	0.024	0.61		90	40	0.0031	0.008	0.20	
35	36	0.0050	0.023	0.60		90	39	0.0035	0.008	0.19	
35	35	0.0056	0.023	0.58		90	38	0.0039	0.007	0.18	
35	34	0.0063	0.022	0.56		90	37	0.0044	0.007	0.17	t
35	33	0.0070	0.021	0.55		90	36	0.0050	0.006	0.15	
35	32	0.0079	0.021	0.52		100	40	0.0031	0.007	0.18	
35	31	0.0089	0.020	0.50	rtp	100	39	0.0035	0.006	0.17	
35	30	0.0100	0.018	0.47		100	38	0.0039	0.006	0.15	
35	29	0.0112	0.017	0.44		100	37	0.0044	0.006	0.14	
35	28	0.0126	0.016	0.40		100	36	0.0050	0.005	0.13	t

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *p.* Nearest size to punched screens, Table II. *t.* Sizes in Table I. *r.* Rittinger's screens.

TABLE VII.—*Sizes of Wire Screens of the Trenton Standard Gauge.*

	Mesh.	Wire. Trenton.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
Inches to the mesh.	3″	0	0.305	2.69	68.3	
	3″	00	0.330	2.67	67.8	r
	3″	000	0.360	2.64	67.1	t
	2½″	1	0.285	2.21	56.1	
	2½″	0	0.305	2.19	55.6	
	2½″	00	0.330	2.17	55.1	f
	2⅛″	0	0.305	1.82	46.2	rt
	1½″	3	0.245	1.25	31.7	t
	1½″	2	0.265	1.23	31.2	r
	1½″	1	0.285	1.21	30.7	
	1⅛″	4	0.225	0.90	22.9	rt
	1″	15	0.070	0.930	23.6	f
	1″	14	0.080	0.920	23.4	
	1″	13	0.092	0.907	23.0	
	1″	12	0.105	0.895	22.7	
	1″	11	0.117	0.882	22.4	
	1″	10	0.130	0.870	22.1	
	1″	9	0.145	0.855	21.7	
	1″	8	0.160	0.840	21.3	
	1″	7	0.175	0.825	20.9	
	1″	6	0.190	0.810	20.6	
	1″	5	0.205	0.795	20.2	
	1″	4	0.225	0.775	19.7	
	1″	3	0.245	0.755	19.2	
	¾″	16	0.061	0.689	17.5	
	¾″	15	0.070	0.680	17.3	
	¾″	14	0.080	0.670	17.0	
	¾″	13	0.092	0.657	16.7	
	¾″	12	0.105	0.645	16.4	
	¾″	11	0.117	0.632	16.0	t
	¾″	10	0.130	0.620	15.7	
	¾″	9	0.145	0.605	15.4	
	¾″	8	0.160	0.590	15.0	
	¾″	7	0.175	0.575	14.6	
	¾″	6	0.190	0.560	14.2	
	¾″	5	0.205	0.545	13 8	
	¾″	4	0.225	0.525	13.3	
	⅝″	17	0.052	0.572	14.5	
	⅝″	16	0.061	0.564	14.3	
	⅝″	15	0.070	0.555	14.1	
	⅝″	14	0.080	0.545	13.8	
	⅝″	13	0.092	0.532	13.5	
	⅝″	12	0.105	0.520	13.2	
	⅝″	11	0.117	0.507	12.9	
	⅝″	10	0.130	0.495	12.6	f
	⅝″	9	0.145	0.480	12.2	
	⅝″	8	0.160	0.465	11.8	r
	⅝″	7	0.175	0.450	11.4	t
	⅝″	6	0.190	0.435	11.0	
	⅝″	5	0.205	0.420	10.7	
Meshes to the inch.	2	18	0.045	0.455	11.56	
	2	17	0.052	0.447	11.35	
	2	16	0.061	0.439	11.15	
	2	15	0.070	0.430	10 92	
	2	14	0.080	0.420	10.67	
	2	13	0.092	0.407	10.34	
	2	12	0.105	0.395	10.03	
	2	11	0.117	0.382	9.70	
	2	10	0.130	0.370	9.40	
	2	9	0.145	0.355	9.02	
	2	8	0.160	0.340	8.64	
	2	7	0.175	0.325	8.25	
	2	6	0.190	0.310	7.87	
	2½	18	0.045	0.355	9.02	
	2½	17	0.052	0.347	8.81	
	2½	16	0.061	0.339	8.61	
	2½	15	0.070	0.330	8.38	
	2½	14	0.080	0.320	8.13	t
	2½	13	0.092	0.307	7.80	
	2½	12	0.105	0.295	7.49	r
	2½	11	0.117	0.282	7.16	
	2½	10	0.130	0.270	6.86	

	Mesh.	Wire. Trenton.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
Meshes to the inch.	2½	9	0.145	0.255	6.48	
	2½	8	0.160	0.240	6.10	
	3	20	0.035	0.298	7.57	
	3	19	0 040	0.293	7.44	
	3	18	0.045	0.288	7 31	
	3	17	0.052	0.281	7.14	
	3	16	0.061	0.272	6.91	
	3	15	0 070	0.263	6.68	
	3	14	0 080	0.253	6.43	f
	3	13	0.092	0.241	6.12	
	3	12	0.105	0.228	5.79	
	3	11	0.117	0 216	5.49	
	3	10	0 130	0.203	5 16	
	3½	21	0.031	0.254	6.45	
	3½	20	0.035	0.250	6.35	
	3½	19	0.040	0.245	6.22	
	3½	18	0.045	0.240	6.10	
	3½	17	0.052	0.232	5.89	
	3½	16	0 061	0.224	5.69	t
	3½	15	0.070	0.215	5.46	
	3½	14	0.080	0.205	5.21	r
	3½	13	0.092	0 192	4 88	
	3½	12	0.105	0.180	1.57	
	3½	11	0.117	0.167	4.24	
	4	22	0.028	0.222	5.64	
	4	21	0 031	0 219	5.56	
	4	20	0.035	0.215	5.46	
	4	19	0.040	0.210	5 33	
	4	18	0.045	0.205	5.21	
	4	17	0 052	0 197	5.00	
	4	16	0.061	0.189	4.80	
	4	15	0 070	0.180	4.57	
	4	14	0 080	0.170	4.32	
	4	13	0.092	0.157	3.99	t
	4	12	0.105	0.145	3.68	
	4	11	0.117	0 132	3.35	
	4½	23	0.025	0.197	5.00	
	4½	22	0.028	0.194	4.93	
	4½	21	0.031	0.191	4.85	
	4½	20	0.035	0 187	4.75	
	4½	19	0.040	0.182	4.62	
	4½	18	0.045	0.177	4.49	
	4½	17	0.052	0.170	4.32	
	4½	16	0.061	0.161	4.09	
	4½	15	0.070	0.152	3.86	r
	4½	14	0.080	0.142	3.61	
	4½	13	0.092	0.130	3.30	
	4½	12	0.105	0.117	2.97	
	5	24	0.022	0.177	4.49	
	5	23	0.025	0.175	4.44	
	5	22	0.028	0.172	4.37	
	5	21	0.031	0.169	4.29	
	5	20	0.035	0.165	4.19	
	5	19	0.040	0.160	4.06	
	5	18	0.045	0.155	3.94	
	5	17	0.052	0.147	3.73	
	5	16	0.061	0.139	3.53	
	5	15	0.070	0.130	3.30	
	5	14	0.080	0.120	3.05	
	5	13	0.092	0.107	2.72	
	6	25	0.020	0.146	3.71	
	6	24	0.022	0.144	3.66	
	6	23	0.025	0.141	3.58	
	6	22	0.028	0.138	3.50	
	6	21	0.031	0.135	3.43	
	6	20	0.035	0.131	3.33	
	6	19	0.040	0.126	3.20	f
	6	18	0.045	0.121	3.07	
	6	17	0.052	0.114	2.89	
	6	16	0.061	0.105	2.67	r
	6	15	0.070	0.096	2.44	
	6	14	0.080	0.086	2.18	

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *p.* Nearest size to punched screens, Table II. *t.* Sizes in Table I. *r.* Rittinger's screens.

TABLE VII.—*Continued.*

Mesh. (Meshes to the inch.)	Wire. Trenton.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
7	26	0.018	0.125	3.17	
7	25	0.020	0.123	3.12	
7	24	0.022	0.120	3.05	
7	23	0.025	0.118	3.00	
7	22	0.028	0.115	2.92	
7	21	0.031	0.112	2.84	t
7	20	0.035	0.108	2.74	
7	19	0.040	0.103	2.61	
7	18	0.045	0.098	2.49	
7	17	0.052	0.090	2.29	
7	16	0.061	0.082	2.08	
7	15	0.070	0.073	1.85	
8	27	0.017	0.108	2.74	
8	26	0.018	0.107	2.72	
8	25	0.020	0.105	2.67	
8	24	0.022	0.102	2.59	
8	23	0.025	0.100	2.54	
8	22	0.028	0.097	2.46	
8	21	0.031	0.094	2.39	
8	20	0.035	0.090	2.29	
8	19	0.040	0.085	2.16	
8	18	0.045	0.080	2.03	r
8	17	0.052	0.072	1.83	
8	16	0.061	0.064	1.62	
9	28	0.016	0.095	2.41	
9	27	0.017	0.094	2.39	
9	26	0.018	0.093	2.36	
9	25	0.020	0.091	2.31	
9	24	0.022	0.089	2.26	
9	23	0.025	0.086	2.18	
9	22	0.028	0.083	2.11	
9	21	0.031	0.080	2.03	t
9	20	0.035	0.076	1.93	
9	19	0.040	0.071	1.80	
9	18	0.045	0.066	1.68	
9	17	0.052	0.059	1.50	
10	29	0.015	0.085	2.16	
10	28	0.016	0.084	2.13	
10	27	0.017	0.083	2.11	
10	26	0.018	0.082	2.08	
10	25	0.020	0.080	2.03	
10	24	0.022	0.078	1.98	
10	23	0.025	0.075	1.90	
10	22	0.028	0.072	1.83	
10	21	0.031	0.069	1.75	
10	20	0.035	0.065	1.65	
10	19	0.040	0.060	1.52	
10	18	0.045	0.055	1.40	
12	30	0.014	0.069	1.75	
12	29	0.015	0.068	1.73	
12	28	0.016	0.067	1.70	
12	27	0.017	0.066	1.68	
12	26	0.018	0.065	1.65	
12	25	0.020	0.063	1.60	f
12	24	0.022	0.061	1.55	
12	23	0.025	0.058	1.47	p
12	22	0.028	0.055	1.40	rt
12	21	0.031	0.052	1.32	
12	20	0.035	0.048	1.22	
12	19	0.040	0.043	1.09	
14	34	0.010	0.061	1.55	
14	33	0.011	0.060	1.52	
14	32	0.012	0.059	1.50	
14	31	0.013	0.058	1.47	
14	30	0.014	0.057	1.45	
14	29	0.015	0.056	1.42	
14	28	0.016	0.055	1.40	
14	27	0.017	0.054	1.37	
14	26	0.018	0.053	1.35	
14	25	0.020	0.051	1.29	
14	24	0.022	0.049	1.24	p
14	23	0.025	0.046	1.17	

Mesh. (Meshes to the inch.)	Wire. Trenton.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference
14	22	0.028	0.043	1.09	
14	21	0.031	0.040	1.02	
14	20	0.035	0.036	0.91	
16	35	0.009	0.053	1.35	
16	34	0.010	0.052	1.32	
16	33	0.011	0.051	1.29	
16	32	0.012	0.050	1.27	
16	31	0.013	0.049	1.24	
16	30	0.014	0.048	1.22	
16	29	0.015	0.047	1.19	
16	28	0.016	0.046	1.17	
16	27	0.017	0.045	1.14	
16	26	0.018	0.044	1.12	
16	25	0.020	0.042	1.07	r
16	24	0.022	0.040	1.02	t
16	23	0.025	0.037	0.94	
16	22	0.028	0.034	0.86	
16	21	0.031	0.031	0.79	
18	36	0.009	0.047	1.19	
18	35	0.0095	0.046	1.17	
18	34	0.010	0.045	1.14	
18	33	0.011	0.044	1.12	
18	32	0.012	0.043	1.09	
18	31	0.013	0.042	1.07	p
18	30	0.014	0.041	1.04	
18	29	0.015	0.040	1.02	
18	28	0.016	0.039	0.99	
18	27	0.017	0.038	0.96	
18	26	0.018	0.037	0.94	
18	25	0.020	0.035	0.89	
18	24	0.022	0.033	0.84	
18	23	0.025	0.030	0.76	
18	22	0.028	0.027	0.68	
20	36	0.009	0.041	1.04	
20	35	0.0095	0.040	1.03	
20	34	0.010	0.040	1.02	
20	33	0.011	0.039	0.99	
20	32	0.012	0.038	0.96	
20	31	0.013	0.037	0.94	
20	30	0.014	0.036	0.91	
20	29	0.015	0.035	0.89	p
20	28	0.016	0.034	0.86	
20	27	0.017	0.033	0.84	
20	26	0.018	0.032	0.81	
20	25	0.020	0.030	0.76	
20	24	0.022	0.027	0.68	
20	23	0.025	0.025	0.63	
22	36	0.009	0.036	0.91	
22	35	0.0095	0.036	0.90	
22	34	0.010	0.035	0.89	
22	33	0.011	0.034	0.86	
22	32	0.012	0.033	0.84	
22	31	0.013	0.032	0.81	
22	30	0.014	0.031	0.79	f
22	29	0.015	0.030	0.76	
22	28	0.016	0.029	0.74	
22	27	0.017	0.028	0.71	
22	26	0.018	0.027	0.69	
22	25	0.020	0.025	0.64	
22	24	0.022	0.023	0.58	
24	36	0.009	0.033	0.84	
24	35	0.0095	0.032	0.81	
24	34	0.010	0.032	0.80	
24	33	0.011	0.031	0.79	
24	32	0.012	0.030	0.76	
24	31	0.013	0.029	0.74	p
24	30	0.014	0.028	0.71	t
24	29	0.015	0.027	0.68	
24	28	0.016	0.026	0.66	r
24	27	0.017	0.025	0.63	
24	26	0.018	0.024	0.61	
24	25	0.020	0.022	0.56	

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″, etc. *p.* Nearest size to punched screens, Table II. *t.* Sizes in Table I. *r.* Rittinger's screens.

TABLE VII.—*Concluded.*

Mesh. (Meshes to the inch.)	Wire. Trenton.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
26	36	0.009	0.029	0.74	
26	35	0.0095	0.029	0.73	
26	34	0.010	0.028	0.71	
26	33	0.011	0.027	0.68	p
26	32	0.012	0.026	0.66	
26	31	0.013	0.025	0.63	
26	30	0.014	0.024	0.61	
26	29	0.015	0.023	0.58	
26	28	0.016	0.022	0.56	
26	27	0.017	0.021	0.53	
26	26	0.018	0.020	0.51	
28	36	0.009	0.027	0.68	
28	35	0.0095	0.026	0.67	
28	34	0.010	0.026	0.66	
28	33	0.011	0.025	0.63	
28	32	0.012	0.024	0.61	p
28	31	0.013	0.023	0.58	
28	30	0 014	0.022	0.56	
28	29	0.015	0.021	0.53	
28	28	0.016	0 020	0.51	
28	27	0.017	0.018	0.48	
30	36	0.009	0.024	0.62	
30	35	0.0095	0.024	0.60	
30	34	0.010	0.023	0.58	
30	33	0.011	0.022	0.56	p
30	32	0.012	0.021	0.53	
30	31	0.013	0 020	0.51	tp
30	30	0.014	0.019	0.48	
30	29	0.015	0.018	0.46	p
30	28	0.016	0.017	0.43	
35	36	0.009	0.019	0.50	
35	35	0.0095	0.019	0.48	
35	34	0.010	0.018	0.46	
35	33	0.011	0.017	0.43	p
35	32	0.012	0.016	0.41	
35	31	0.013	0.015	0.38	rpf
35	30	0.014	0.014	0.35	
35	29	0.015	0.013	0.33	
40	37	0.0085	0.016	0.42	
40	36	0 009	0.016	0.41	
40	35	0.0095	0.015	0.40	
40	34	0.010	0.015	0.38	
40	33	0.011	0.014	0.35	tp
40	32	0.012	0.013	0.33	
40	31	0.013	0.012	0.30	
45	37	0.0085	0.014	0.35	
45	36	0.009	0.013	0.34	
45	35	0.0095	0.013	0.32	

Mesh. (Meshes to the inch.)	Wire. Trenton.	Wire. Diameter in Inches.	Space. Inches.	Space. Milli-meters.	Reference.
45	34	0.010	0.012	0.30	
45	33	0.011	0.011	0.28	
45	32	0.012	0.010	0.25	
50	37	0.0085	0.011	0.29	
50	36	0.009	0.011	0.28	
50	35	0.0095	0.010	0.27	
50	34	0.010	0.010	0.25	rt
50	33	0.011	0.009	0.23	
50	32	0.012	0.008	0.20	
55	38	0.008	0.010	0.25	
55	37	0.0085	0.010	0.24	
55	36	0.009	0.009	0.23	
55	35	0.0095	0.009	0.22	
55	34	0.010	0.008	0.20	
60	38	0.008	0.009	0.22	
60	37	0.0085	0.008	0.20	
60	36	0.009	0.008	0.19	f
60	35	0.0095	0.007	0.18	
60	34	0.010	0.007	0.16	
64	38	0.008	0.008	0.19	
64	37	0.0085	0.007	0.18	
64	36	0.009	0 007	0.17	t
64	35	0.0095	0.006	0.16	
64	34	0.010	0.006	0.15	
70	39	0.0075	0.007	0.17	
70	38	0.008	0.006	0.15	r
70	37	0.0085	0.006	0.14	
70	36	0.009	0.005	0.13	
70	35	0.0095	0.005	0.12	
74	39	0.0075	0.006	0.15	
74	38	0.0080	0.005	0.14	
74	37	0.0085	0.005	0.13	
74	36	0.0090	0.004	0.12	
74	35	0.0095	0.004	0.10	
80	39	0.0075	0.005	0.13	
80	38	0.0080	0 004	0.12	t
80	37	0.0085	0.004	0.10	
80	36	0.0090	0.003	0.09	
80	35	0.0095	0.003	0.08	
90	40	0 0070	0.004	0.10	
90	39	0.0075	0.004	0.09	tf
90	38	0.0080	0.003	0.08	
90	37	0.0085	0.003	0.07	
90	36	0.0090	0 002	0.05	
100	40	0.0070	0 003	0.08	
100	39	0.0075	0.003	0.07	t
100	38	0.0080	0.002	0.05	
100	37	0.0085	0.001	0.04	

f. Nearest approach to sizes of graded set, 1″, ½″, ¼″. etc. *p.* Nearest size to punched screens, Table II. *t.* Sizes in Table I. *r.* Rittinger's screens.

The Ancient Copper-Mines of Lake Superior.

BY ALVINUS BROWN WOOD, DETROIT, MICH.

(Bethlehem Meeting, February, 1906.)

THE ancient copper-mines of Lake Superior, having been destroyed or covered by modern mining-dumps, are not accessible to the present inhabitants of that region, and, since no more are likely to be found, some account of those already discovered, from one who has seen them, may be acceptable to the Institute.

Early in May, 1855, I took a party of men to open an ancient pit on the Quincy mine-location. Mr. C. C. Douglas was the "Agent," as mine-managers were then called on Lake Superior. The Quincy Co. had previously done work on a fissure-vein, several hundred feet west of the Amygdaloid lode (afterwards called the Pewabic lode), on which was the ancient pit herein described. It was at this pit that the Quincy main shaft was subsequently sunk.

The ancient pit showed an apparent depth of about 4 ft., but when cleaned out it was shown to be 14 ft. deep, having been filled up by the sliding-in of material composed largely of broken rock, taken out in sinking the ancient shaft, and left near its mouth. Midway of the shaft, and below water-level, was found a scaffolding of white-birch poles, which had been used as a landing, to assist in removing the excavated rock. The shaft was 7 ft. in diameter. The excavation had been done by first building fires in order to disintegrate the rock, as was proved by the abundance of charcoal found; then breaking the softened rock with hammers of diorite, such as is found in the greenstone range on Keweenaw point. These hammers had about the size of large paving-stones. Only broken ones were found, the whole ones having been doubtless carried away for other work, such as could be traced on the lode for a hundred feet or more to the south.

At the place where this shaft was sunk, barrel-copper (*i.e.*,

copper too coarse to go to the mill, yet not in pieces large enough to be shipped separately) was abundant. This class of copper was most valuable to the old miners; since large masses were too large for them to handle, and stamp-copper was too small to serve their uses. It is evident that they did not understand the art of smelting.

This excavation represents the removal of 41 tons of rock, and the ground at that place, judging from what could be seen in the walls of the shaft, carried fully 12 per cent. of coarse copper. This would represent 5 tons of copper actually extracted. From the amount of ancient work showing further south on the lode, it would not be too liberal to estimate that four times as much more copper, or, say, 25 tons in all, was obtained from this locality. The amount may have been much greater, for the ground under the workings was afterwards proved to be very rich in barrel-copper.

I still have in my possession a handful of fragments of copper, torn from the barrel-copper by blows from the stone hammers, each fragment showing the torn face caused by the blow. These were found in the bottom of the shaft.

At about the same time, Mr. Charles H. Palmer, Agent, and Mr. Wm. B. Frue, his foreman, opened an ancient pit on the Pewabic mine location, where the Pewabic main shaft was afterward sunk. The lode at that place was very rich in barrel-copper for a width of 20 ft. or more. The pit was somewhat irregular, about 6 ft. deep and 15 ft. wide, the length being 20 ft. across the lode, and representing the removal of 138 tons of rock, which carried about 10 per cent. of barrel-copper, or 13.5 tons of metallic copper.

The ancient work extended to the south more than 100 ft., but was not so deep. The ground under these pits proved to be very rich in barrel-copper, as is shown by the workings of the Pewabic Co. (This ground now belongs to the Quincy Co.) I think it safe to estimate the copper taken from this south working by the ancients to be twice that taken from the shaft, or a total of 40.5 tons—probably a great deal more. In the bottom of this shaft were found the remains of a large fire, the unconsumed brands of which had not been moved, indicating a sudden stoppage of the work. One brand, a part of a young black-oak tree, 6 in. in diameter, showed, by the form

of the cutting, that it had been cut at the stump while standing. Having been continuously under water, it was well preserved and of full form. It had evidently been hewed with a thin-edged tool, about 2 in. wide. A thin chip, which had been cut from the face left by a previous chip, was broken off at the base, about $\frac{1}{8}$ in. thick; but the cutting-tool had gone down 0.25 in. below the upper end of the remaining stub, leaving a gash into which I inserted my knife-blade. This proved that the cutting had been done by a thin, sharp blade, probably of copper, and not by a stone axe.

On the Franklin location, further north, the surface material was deeper, and no ancient work was found.

The ancient miners worked only where the "wash" covering the cropping of lodes or veins was thin, and the lodes were easily found.

Next to the north is the Mesnard mine location. No rich ground was found there, such as the ancients would mine. But near the south boundary, about 200 ft. west of the line of the Pewabic lode, was found a large, broad ancient pit that did not reach bedrock. Around it a cart-load of broken stone hammers might have been gathered, some weighing as much as 20 lb.; none but broken ones remained, the sound ones having been taken away, doubtless for other work. When I discovered the pit, I thought I had made a discovery of value, but the work done showed its true character.

These hammers had probably been used to work up a mass of float-copper of several tons weight, similar to the Worminghaus mass found on the Mesnard location, about 1,000 ft. to the northeast, and east of the Pewabic lode. This mass was rocky, and could have been broken up with stone hammers; its weight was about 15 tons, as I remember. It had escaped the notice of the ancient miners.

On the south side of Portage lake, the Isle Royale lode had been extensively worked on the Isle Royale location, by the ancient miners. The Isle Royale shaft was sunk on the main old workings. I did not see this ancient work, since it was covered with waste dumps from the mine when I first visited the mines, in 1854.

At the Huron mine, next south on the Isle Royale lode, the first four shafts were started in 1854, in ground not suited to

the ancient miners. In 1856 I took charge of the property, as agent, and pursued work in the first shafts. In 1857 I discovered signs of ancient workings to the south; but in this work, instead of digging pits, the lode had been stripped from north to south. The large stones had been piled up, and the dirt thrown back as the work progressed.

I caused the lode to be cleaned off, beginning at the north end of the old work, for a length of 30 ft. and a width of 16 ft. To the south there were abundant signs of ancient work for an additional distance of 80 ft. About 3 ft. in depth of the lode had been removed where it was uncovered. This surface had had the benefit of disintegration since the Ice Age.

To get an idea of the amount of copper which the ancients obtained, I had the uncovered surface—30 by 16 ft.—worked over with pick and bar, no powder being used. Ten tons of barrel-copper were taken up; the largest piece weighing 1,500 lb. The copper obtained was weighed and shipped to the copper smelting-works at Detroit. I should estimate that the ancients got twice as much as we did, or 20 tons, to say nothing of what they took from the additional ground worked by them to the south.

In the deepest part of this old work, in a pit about 1.5 ft. below the general level of the bottom, I found a leather string about 30 in. long, which was strong and in good condition, having lain under water. About 20 in. of it is buckskin, the balance being beaver-skin, wrapped with colored porcupine-quills. This, and a small buckskin bag, found in an ancient pit near Rockland, Ontonagon district, are the only relics of a personal nature ever found in the ancient mines, so far as I am informed.

No signs of graves, or of foundations to structures, have been found. Apparently the ancient miners visited this region in the summer season only; and probably they came by the Lakes. That they were navigators is shown by their presence on Isle Royale.

North of the Mesnard mine, the first ancient work was found at the Calumet mine—a single pit, sunk on the Calumet conglomerate. It was discovered by Amos Scott, a man whom I knew well. He was employed by Ed. Hulbert, as foreman, on a property east of the Calumet mine, and called, at that time,

the Hulbert mine. The copper was abundant where the ancient pit was sunk, but, being of stamp-copper size, it did not suit the old miners' purposes as well as the barrel-copper found at other places; and this accounts, no doubt, for the limited amount of work done there. The discovery of this pit led Hulbert to purchase from the St. Mary's Mineral Land Co. the lands of the Calumet and Hecla mines.

The next place further north, where ancient work led to the discovery of a mine, was at the Cliff mine, near Eagle river, which was famous and profitable in its day. The vein produced mass-copper.

The next mine of value to the east was the Central mine, located upon the discovery of an ancient pit on the vein, in the bottom of which appeared a large mass of copper. This mass had been well pounded, to detach such pieces of copper as could be thus secured; but the main body was too large to be handled by the ancients.

Still further east, the old North West mine, later known as the Pennsylvania and the Delaware mines, was discovered at a very early day through an ancient pit on the fissure-vein, near the greenstone range.

North of this range, and nearly north of the North West mine, a number of ancient pits were found on a small fissure-vein, which carried sheet-copper well suited to the uses of the old miners. These pits were discovered in 1853, and were frequently seen by me while they were being cleaned out. The stone hammers used at this place were mostly encircled by well-marked grooves; whereas, at the other ancient mines described, the stone hammers found were not grooved, and only a few of them had their sharp corners rounded. Several cedar shovels, or scrapers, about 3.5 ft. long, with the blade on one side only, were found in these pits.

Extensive ancient mining was done on the easterly part of Isle Royale, on narrow veins yielding sheet-copper. This form being, as already observed, suited to the needs of the old miners, they did much work on these veins.

About 1872, Mr. A. C. Davis worked, on the island, a property called the Minong, and brought to Detroit a very interesting piece of ancient work—a mass of copper of 4 or 5 tons, taken from the bottom of an ancient trench along a vein. This

mass had been deprived of all portions of copper that could be detached by pounding with stone hammers, and the whole surface of the mass had been pounded into deep pits and ridges in attempts to detach other pieces. It showed an immense amount of labor done. Mr. John R. Grout, of the Detroit copper-smelting works, shipped the mass to the Michigan University at Ann Arbor, and offered to sell it to the University at half its cash value, but the Regents did not make the purchase. Perhaps they lacked money; perhaps they lacked appreciation. The mass was shipped back to the smelting-works and went into the furnace. Thus a thing of very great interest was lost. Had the University taken it, it would be the most interesting in all its collections, and of special interest to Michigan.

In the Ontonagon region, extensive ancient mining-work was done at the Minnesota mine, in the fissure-vein, where the historic mass of copper was found below water-level, shored up on timbers, in an ancient shaft, the work having been continued below the mass.

In the Rockland district, extensive work was done on the Amygdaloid lodes, where rich bunches of barrel-copper were found. I was not personally acquainted with these workings.

In 1870 there lived at Gratiot lake, in Keweenaw county, Michigan, a gentleman 70 years of age, of good character and intelligence, who told me that he was in the Ontonagon district at the beginning of mining there, and was conversant with the old miners' work. He said that, at a point on one of the Rockland amygdaloid lodes, where barrel-copper was abundant, an ancient drift had been driven for 40 ft.; at the inner end, in the upper dry corner, a considerable roll of white birch-bark was found; which, on being unrolled, was seen to be covered with what he called "hieroglyphics." He said he saw this roll, and that it "was carried East by a man," and he never heard anything more of it.

The native copper of the Lake Superior mines is accompanied, to a limited extent, by native silver, attached to the surface of the copper, and, at times, inclosed within it, the two being firmly united and each pure, and in no case alloyed. This condition has been considered peculiar to the Lake Superior mines, and as proof that specimens exhibiting it came

from Lake Superior. Samples of this kind, found in the ancient mounds of Ohio, have consequently connected the mound-builders with the old miners. The mound-builders, it is thought by historians, migrated south 900 or 1,000 years ago, and were the Toltecs or the Aztecs of Mexico, probably the latter. This gives some indication as to the time when the ancient mining-work was done.

The copper they obtained by working the ancient mines, as shown by the partial estimates on previous pages, must have amounted to several hundred tons. They also had the first picking of the float-copper, dislodged from the lodes and veins during the Ice Age, which must have amounted to many tons. The present inhabitants have picked up many thousands of pounds. The wonder is that the ancients did not do more mining, since the copper, being so greatly superior to their stone implements, must have been of great value to them. The limited amount of work done by them may indicate that they occupied the country but a short time.

More copper implements have been found in the ancient mound-country of Wisconsin than in any other State, as is shown by the collection at the State University at Madison. Illinois, Michigan and Ohio have yielded many specimens of implements. The Field Museum at Chicago has many valuable specimens from these States.

It must be remembered that all the implements thus found were lost treasures of the utmost value to the owners. I have a spear-head, 11 in. long, of very perfect construction, which was taken from a mound on the left bank of the Wisconsin river, just below the Dalles. This may, or may not, have been a burial mound.

All of the older valuable mines of the Portage Lake district, as well as those of the Keweenaw and the Ontonagon districts, were found by means of the ancient workings, which are now destroyed or covered by mine-dumps.

The new mines, opened in the past few years on the south side of Portage lake, were covered too deep by soil and glacial drift to be found by the ancients; and the same may be said of those opened on the Osceola and the Kearsarge lodes on the north side of Portage lake.

Nine years ago I purchased from a curio dealer at Denver a

thin circular disk of native copper and silver, nearly one-half silver. The origin or source of the metals from which it is made is evidently the Lake Superior mines. This disk, shown in Fig. 1, is 2.5 by 3.5 in. in size; a projection on one side serves for a handle, a hole at the end being doubtless modern. The disk is concave, and on the concave side there is a conventional picture of the face of the sun, outlined by deep creasings with a sharp point. The face is 1.5 in. in diameter, and not quite perfectly round. There are 13 radiating curved lines drawn from the face to the margin, which may represent the 13 lunar months. The lines show on the back as raised lines. The eyes have that straight horizontal appearance to be seen in the ancient carvings on stone, which are exhibited in the Mexican National Museum in the City of Mexico.

I also bought a rude bracelet of the same material, roughly pounded out and bent to spring on to the wrist. It is very irregular in form; from 0.6 to 1.5 in. wide, with very jagged edges. On it, for ornament, are placed four turquoises of excellent quality, but roughly ground into faces. Turquoise, used to ornament a bust carved in stone, is to be seen in the Mexican Museum.

The turquoise is a Mexican stone, not having been mined in United States territory until recently, except at the ancient turquoise mine near Santa Fé, in New Mexico. The stones are set on to the bracelet by cutting four clips in the copper for each stone, and raising these clips, or tags, so as to inclose the stones and keep them in place.

The dealer of whom I bought these articles told me that he bought them in the City of Mexico, from a very large collection of minerals and ancient relics that had been collected by several generations of owners. The owner at that time, not being so interested as his ancestors, was selling off the collection as he had opportunity. The sun-disk, being of a peculiar combination of metals, and a religious emblem as well to the old sun-worshipers, would be likely to be handed down through many centuries by the ancient people. The bracelet may have come down with it. It would have been highly prized for its unusual material, and may have been regarded as a sacred thing. The material for both disk and bracelet being of undoubted Lake Superior origin, raises the question of how it got to Mexico in

the olden time, and the ready reply is,—that it was probably taken there by the old mound-building miners, the Toltecs or Aztecs, when they migrated to the south. They were probably driven from home by an enemy, for in the Lake region they were in a good country, desirable to live in.

With the facts given herein, and if the inferences stated are correct, then we have a clear indication as to who did the ancient mining on Lake Superior, and also as to who were the

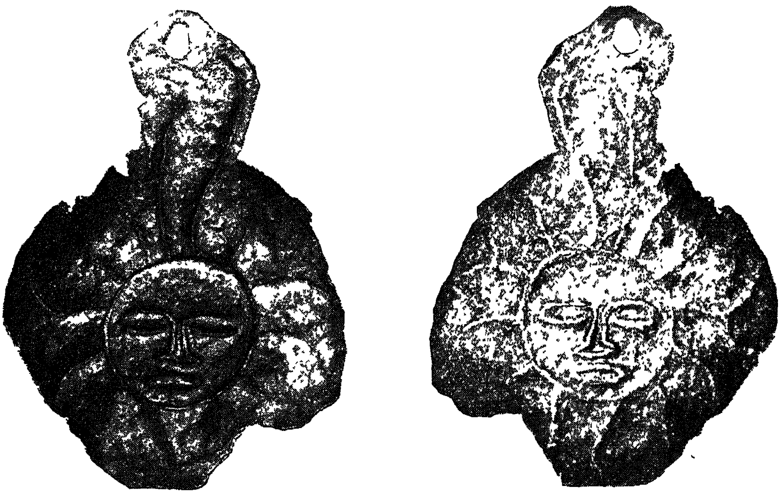

FIG. 1.—SUN-DISK OF NATIVE COPPER AND SILVER.

mound-builders, namely, the Toltecs or the later Aztecs of Central Mexico, probably the latter.

Fig. 1 is a three-quarter sized photographic view of the sun-disk. The lighter shades represent the silver portion and the darker shades the copper. The back, or convex side, shows most silver.

At the first visit of the old French missionaries to Lake Superior, the Indians knew nothing of the old copper-mines, though they had lumps of float-copper, which they regarded as "Manitou." Nor did the modern Indians know anything of the old mines.

The Secondary Enrichment of Copper-Iron Sulphides.*

BY THOMAS T. READ, NEW YORK, N. Y.

(Bethlehem Meeting, February, 1906)

THE fact that certain types of ore-deposits have attained their present condition through the action of descending surface waters was, perhaps, first clearly pointed out by Posepny.[1] The oxidizing effects of such waters were discussed by Penrose,[2] and many papers dealing with this phase of the subject have subsequently appeared. That secondary enrichment of sulphide ores, however, might be produced by descending surface waters was not clearly understood until after the copper-bearing veins of the Butte district, Mont., had been carefully studied by the geologists of the U. S. Geological Survey, although De Launay[3] had suggested that sulphides might be transported from point to point and re-deposited in the zone above the level of ground-water. As a result of the study of the Butte district, two papers by geologists engaged in the work appeared within a short interval. The first by Weed,[4] dealing with the enrichment of veins by later metallic sulphides; and the second by Emmons,[5] treating of the secondary enrichment of ore-deposits in general. In the latter paper it was pointed out that a typical vein of sulphide ores which has been subjected to the action of descending surface-waters exhibits four zones,—an upper or surface-zone, in which the changes have been mainly of removal, the products being typically oxides; a second zone of oxide enrichment, in which the less soluble metals brought from above have been precipitated as carbonates or oxides; a third zone of sulphide enrichment,

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[1] *Trans.*, xxiii., 197–369 (1893).

[2] *Journal of Geology*, vol. ii., pp. 288–317 (1894).

[3] *Annales des Mines*, vol. xii., pp. 119–228 (1897).

[4] *Bulletin of the Geological Society of America*, vol. xi., pp. 179–206 (1900).

[5] *Trans.*, xxx., 177–216 (1900).

in which the materials brought down past the second zone have been deposited as sulphides; and, finally, the zone of unaltered sulphides.

In both these papers, various suggestions were made concerning the probable chemistry of the process. Somewhat later Winchell[6] detailed the results of his experiments with the effect of solutions of $CuSO_4$ and SO_2 upon lean sulphides, and pointed out their bearing upon the production of rich sulphides, such as chalcocite, from sulphides poor in copper. Still more recently, Kemp[7] has pointed out that, in very many instances, lean sulphides are later than the rich sulphides with which they are associated, or that there may be lean sulphides which are younger and others which are older than the rich ores. In other words, the changes which have taken place are reversible, an observation which is very valuable.

In treating the chemistry of such secondary enrichment, all the foregoing writers have assumed that chalcopyrite has the formula Cu_2S, Fe_2S_3. Some years ago, Morgan and Smith[8] demonstrated that the iron of chalcopyrite is all ferrous, or, conversely, that its copper is cupric and its formula, therefore, is CuS, FeS. Obviously, this must radically affect the reactions into which chalcopyrite may enter. In the paper by Weed, already cited, little attention is paid to the valency of the copper, and copper-glance is frequently referred to as cupric sulphide. On the other hand, a writer under the pseudonym of Agricola[9] has recently criticised a reaction suggested by me for the simultaneous production of covellite and chalcocite from chalcopyrite, on the ground that the cuprous copper of the chalcopyrite could not yield cupric sulphide (covellite) in the presence of SO_2. Since the copper of chalcopyrite is cupric, this criticism is without point.

The experimental work of Winchell, already mentioned, has been so widely quoted and so extensively drawn on by others, that it seemed advisable to undertake further work along the same lines in order to elaborate more clearly, if possible, the actual processes involved in the secondary enrichment of cop-

[6] *Bulletin of the Geological Society of America*, vol. xiv., p. 269 (1903).
[7] *Economic Geology*, pp. 11–26 (1905).
[8] *Journal of the American Chemical Society*, vol xxiii., p. 107 (1901).
[9] *Engineering and Mining Journal*, vol. lxxix., p. 1147 (1905).

per-iron sulphide ores. The method followed by Winchell was to place pyrite, containing a small percentage of copper, in a sealed jar with a solution of $CuSO_4$ in one case, and of $CuSO_4$ plus SO_2 in the other. After the lapse of a considerable interval of time no effect was apparent in the jar containing $CuSO_4$ only, but in that containing SO_2 as well, the grains of pyrite had become covered with a blue-black coating resembling chalcocite. From these results and some other chemical experiments it was inferred that $CuSO_4$ was without action on lean sulphides, and that the SO_2 reduced the $CuSO_4$ to Cu_2SO_4, which then reacted on the pyrite to form $FeSO_4$, Cu_2S and S.

In conducting further experimental work, an attempt was made to imitate natural conditions as nearly as possible. Coarsely-crushed pyrite and chalcopyrite were placed in open glass tubes, and a solution of $CuSO_4$ was allowed to percolate slowly through one and a solution of $CuSO_4$ plus SO_2 through the other; 100 cc. of a 1.5-per cent. solution was used in each case, being passed through from 15 to 20 times in the course of two weeks. At the end of that time the solutions were analyzed, which showed that, in the case of the $CuSO_4$ plus SO_2, the percentage of copper present had decreased, and a notable amount of the iron had gone into solution as $FeSO_4$. In the case of the $CuSO_4$ alone, a small amount of iron had gone into solution. In using this method, it proved so difficult to prevent various losses, as well as to guard the ore and solutions from the effects of vagrant gases in the laboratory, that it was abandoned, and a study of the reactional changes in closed vessels was substituted in its place. Small (150-cc.) bottles were used for this purpose. For the original copper-iron sulphide, chalcopyrite, crushed between 40 and 100 mesh, was selected, and in order that the enrichment changes might have opportunity partly to establish themselves, it was warmed with dilute $CuSO_4$ solution until it became distinctly tarnished, and then washed and dried. An analysis showed it to contain Cu, 32.62, and Fe, 27.24 per cent. Ten grams of this material were placed in each bottle, and 100 cc. of each of the following solutions were added to the corresponding bottle:

1. A solution of copper sulphate containing 3.6 mg. of copper per cc., saturated with SO_2 gas.

2. A solution of copper sulphate containing 3.6 mg. of copper per cc.

3. Distilled water saturated with SO_2 gas.

4. A solution of calcium bicarbonate.

The bottles were closely stoppered with rubber stoppers, which inclosed about 40 cc. of air above the level of the solution. They were shaken daily, and allowed to stand for a month.

At the end of the month the solutions were filtered off, the residue washed, dried and weighed, and both were analyzed. The results obtained were as follows:—

1. *Copper Sulphate and Sulphur Dioxide Gas.*—The loss in weight of the copper-iron sulphide was 4.6 per cent. The composition of the enriched sulphide was: Fe, 26.86 per cent., a loss of 0.38 per cent.; and Cu, 33.90 per cent., a gain of 1.28 per cent. The composition of the solution was: Fe, 3.43 mg. per cc., a gain of 3.43; Cu, 3.38 mg. per cc., a loss of 0.22; and SO_3, 9.91, a gain of 5.38 mg. per cc.

The enriched sulphide was dark green in color; during the month it had been successively bronzy, purple, and dark steely blue. A few brassy grains could be seen.

2. *Copper Sulphate Alone.*—The composition of enriched sulphide was: Fe, 27.30 per cent., a gain of 0.06 per cent.; and Cu, 33.77 per cent., a gain of 1.15 per cent. The composition of the solution was: Fe, 0.1 mg. per cc., a gain of 0.1; Cu, 3.25 mg. per cc., a loss of 0.35; and SO_3, 5.1 mg. per cc., a gain of 0.57.

The color of the sulphide had not appreciably changed.

3. *Water and Sulphur Dioxide.*—The loss in weight of the copper-iron sulphide was 5.04 per cent. The composition of enriched sulphide was: Fe, 26.95 per cent., a loss of 0.29 per cent.; and Cu, 34.06 per cent., a gain of 1.44 per cent. The composition of the solution was: Fe, 2.67 mg. per cc., a gain of 2.67; Cu, 0.06 mg. per cc., a gain of 0.06; SO_3, 3.81 mg. per cc., a gain of 3.81.

The sulphide was a lustrous blue, almost exactly like covellite, although scattered purple and brassy grains could be detected.

4. *Calcium Bicarbonate.*—The calcium bicarbonate solution was prepared by treating the dust of a very pure marble with

ordinary carbonated or with "vichy" water, filtering, and adding carbonated water until the cloudiness disappeared. The strength of the solution so prepared was 0.15 mg. of CaO per cc. The copper-iron sulphide showed a loss in weight of 0.1 per cent. It had not changed appreciably in appearance. The solution contained 0.36 mg. of SO_3 per cc. and very small amounts of iron and copper; but, unfortunately, it was not possible to make their determination until after some CO_2 had escaped, causing a precipitation of $CaCO_3$ and a consequent alteration of the composition of the solution.

The solid residue effervesced slightly with hydrochloric acid, showing a partial change to carbonates. It was plain that, to some extent, the calcium bicarbonate had formed calcium sulphate and carbonates of iron and copper.

In considering the foregoing results, the most important point of interest is that the solution containing SO_2 alone had attacked the copper-iron sulphide, with the resultant formation of $FeSO_4$, the amount of $CuSO_4$ formed being very small, indeed. There is no free sulphuric acid present. Whether the sulphide acts as a catalyzer, oxidizing the sulphurous to sulphuric acid, which immediately reacts with the sulphide, or whether the sulphurous acid attacks the sulphide, forming sulphites, which are then oxidized to sulphates by the dissolved air, is not evident. The important fact is clear, however, that waters containing SO_2 and O will attack copper-iron sulphides, abstracting the iron and removing it in the form of $FeSO_4$, the enrichment of the sulphide being due to the removal of its iron.

In the case in which $CuSO_4$ and SO_2 were used, a similar action had taken place, but the removal of the iron and production of $FeSO_4$ had been greater. The solution also showed a loss in copper, so that, evidently, the $CuSO_4$ had reacted with the copper-iron sulphide, exchanging its copper for the iron of the sulphide.

In the case of the $CuSO_4$ alone, copper was removed from the solution and the copper-iron sulphide was enriched in copper. But there was very little $FeSO_4$ produced, the action apparently being a reduction of the $CuSO_4$ by the sulphide and the production of free sulphuric acid. Why the free sulphuric acid does not attack the sulphide is puzzling, and how, under

the circumstances, there could be a gain in the SO_3 is no less so. The difficulty of obtaining a uniform mixture of this coarsely crushed material was great.

In the case of the calcium bicarbonate solution the action was very slight.

An important factor in the enrichment of copper-iron sulphides is the removal of the iron, either in solution as sulphate or by simple oxidation. This is also seen by a study of the veins themselves. Weed[10] mentions that, at Gold Hill, N. C., the oxidized ore is bordered by massive copper-glance, and, in numerous other localities, this phenomenon is less clearly shown. Very similar is the action of kernel-roasting of low-grade copper-iron sulphides, when, by oxidation of the iron, an enriched core is produced, resulting, finally, in the production of a button of metal, *i.e.*, copper, at the center, if the action is pushed to its limit. The oxidizing action being slow, the copper retreats toward the center as the iron is oxidized. Just how this is accomplished is not known, but it seems to be a slow molecular flow, which, like the formation of concretions, is not yet clearly understood.

The most important feature of the foregoing is the demonstration that it is unnecessary to provide an hypothesis to explain the reduction of copper from bivalency to monovalency, during the process of enrichment. The auto-reduction of the copper by the successive removal of the iron and sulphur by oxidation is quite similar to that which takes place in the converting of a low-grade matte. Indeed, Cu_2S, compared with Cu_2S_2, is a partly oxidized salt, for the latter will require twice as much oxygen as the former, in order to form metallic copper. It is clear, then, that oxidation alone is sufficient to produce the enriched sulphides occurring in the zone of secondary enrichment. That the removal of the iron and sulphur has usually sufficed for this enrichment, is not to be believed, since the diminution of volume must then produce very porous ore-bodies. That the retreat of the copper before the advance of the oxidizing zone can carry it to any great distance has yet to be demonstrated. Probably removal by simple oxidation and reaction with descending sulphates have both operated, perhaps

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[10] *Bulletin of the Geological Society of America*, vol. xi., p. 191 (1900).

simultaneously. It cannot be doubted that the processes and actions involved in the production of such ore-bodies have not only varied somewhat in different regions, but also in each individual case, since the conditions change from time to time. The degree of concentration of the solution, the potent influence of very small quantities of dissolved salts in affecting the action of other salts in the same solution, the effect of mass,—all these complicate the problem.

Excluding the ore-deposits of igneous origin, in the present state of our knowledge it seems clear that ore-bodies have been formed by the slow action of dilute solutions. In some cases, these dilute solutions have been the waters escaping from the solidification of igneous intrusives. In others, vadose waters have removed the scattered metallic contents of the rocks, only to concentrate them in a vein. Finally, descending waters may remove the metallic contents of the upper part of an ore-deposit, concentrating them at lower levels. The infinitude of possible varieties of products and reactions has contributed to make the study of ore-deposits a puzzling one indeed, and the greatest credit is due to the economic geologists and mining engineers who, by their keen observation and sagacious inference, have contributed so much to the elucidation of this subject.

The Mining, Preparation and Smelting of Virginia Zinc-Ores.

BY THOMAS LEONARD WATSON, VIRGINIA POLYTECHNIC INSTITUTE, BLACKSBURG, VA.

(Bethlehem Meeting, February, 1906.)

INTRODUCTION.

IN a paper read by title at the Washington meeting of the Institute,[1] May, 1905, I discussed at considerable length the geological relations, mode of occurrence, and the genesis of the lead- and zinc-ores of the Virginia-Tennessee region. The present paper treats of the purely economic phases of the zinc-ores of the district,—such as methods of mining the ores, their preparation and treatment. The Albemarle county mines and plant were discussed in the paper referred to above, and will not be considered here.

METHODS OF MINING.

Until very recently the mining of lead- and zinc-ores in Virginia was limited wholly to the surface-belt of weathering. The mode of occurrence and the character of the ores to be mined were such as to demand the employment of methods used in mining secondary concentrated ores. Accordingly, a system similar to that practiced in mining brown iron-ores was employed, being modified to meet the changed conditions.

Oxidized or Soft Ores.—Up to the present time practically all the ore mined in the Virginia district has been of the oxidized or soft type;—the silicate, calamine, and the carbonate, smithsonite; largely the former, which has been localized and concentrated at only slight depths below the surface. Because of the extreme irregularity with which the limestone weathers when stripped of the overlying residual clay, it presents a roughened surface of large and small irregular "chimneys"

[1] Lead- and Zinc-Deposits of the Virginia-Tennessee Region, *Trans.*, xxxvi., 681 to 737.

and pinnacles of the hard rock, as shown in Figs. 1 and 2. It is in the depressions between the pinnacles, and extending some distance up and on their sides, occasionally passing over the pinnacles from bottom to top, that the ores have been concentrated. The ores hug somewhat closely the irregular surface of the limestone and are overlain by the clay. In this mode of occurrence the ore varies from 1 to 25 ft. in thickness, and its depth below the surface will not exceed 80 or 90 ft., usually much less.

Such an occurrence favored the working of the ores in the beginning by the method of "open-cut" mining, the usual method practiced in the region for mining the brown iron-ores

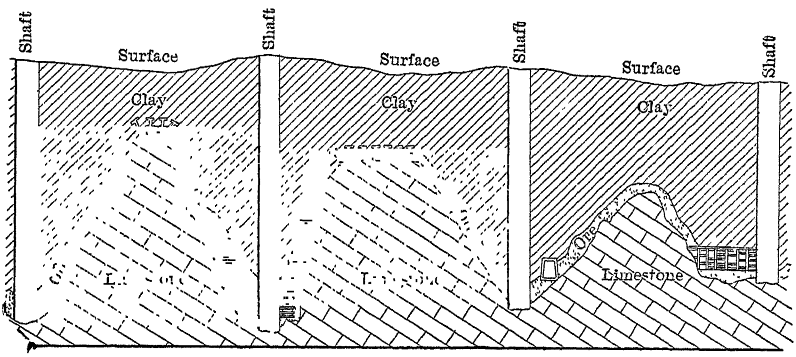

FIG. 1.—SECTION THROUGH THE BERTHA ZINC MINES, SHOWING OCCURRENCE OF ZINC-ORE IN RELATION TO THE LIMESTONE AND CLAY, AND THE METHOD FORMERLY EMPLOYED IN MINING THE ORE. (Modified from Fig. 7 of Mr. Case's paper, *Trans.*, xxii., 524.)

(limonite). The open-cut method of mining the zinc-ores differs, however, from that of mining the iron-ores, in that the former is one essentially of stripping, while in the latter the ores and inclosing clays are mined together and sent to the washer for separation.

At Bertha, the most systematically worked of the Virginia soft-ore zinc-mines, the method of open-cut mining was followed until the fall of 1889, when a change was made. The difficulties of the open-cut mining at Bertha were numerous, and, as the work progressed, these, added to the increased cost of removing the clay, necessitated a change in the method. The chief object, however, in making this change was to secure a steady and sufficient output for the furnaces, as well as to lessen

the cost of mining. Accordingly, a system of underground mining was introduced, the ore being reached through shafts 3.5 ft. in diameter sunk in the clay. Inexpensive plank shafts of 38 in. square inside dimension were also used. These were sunk to the limestone bottom at the deepest points. Timbered drifts were then driven in the ore, following it around the "chimneys." When the chimneys were thus encircled at a given level, a second drift was established or run upon the ore below. In this way drifts, one after another, were carried around and over the chimney until all the ore had been removed from it. This method of mining is illustrated in Fig. 1, which is a modification of Fig. 7 in the paper by Mr. William H. Case.[2] Fig. 2 is a photographic view of the "chimneys" at Ivanhoe, Va. The ore was wheeled to the shafts, where it was loaded into iron buckets and hoisted by steam-power. It was then dumped into cars of about 2 tons capacity, and carried by a locomotive to the dumps at the head of the water-carriage. The ore, as thus mined, was reported to have contained about 26 per cent. of metallic zinc.

On the Lead Mine tract at Austinville, 10 miles west of Bertha, lead- and zinc-ores above ground-water level have been mined for 150 years. Large quantities of the ores have been removed, mined principally by the open-cut method. Much underground mining was done above ground-water level, consisting of shafts, drifts and stopes, reaching a maximum depth of 250 ft. Both the open-cut and underground mining have been done without any apparent system. The soft lead-ores were mined on this tract for a long period of years before attention was given to the zinc-ores, if, indeed, they were recognized at all.

The method used in mining the lead-ore seems to have been that of following the ore, by irregular underground openings, down from the surface through the clays wherever found, resembling a very crude form of stoping. Ordinarily, little or no timbering was done, regard being had only for the present mining of the ore.

The Sulphide Ores.—Practically no sulphide ores have been mined, and, apart from prospecting and development-work,

[2] The Bertha Zinc-Mines, *Trans.*, xxii., 511–536 (1893).

practically no underground mining in the hard rock, limestone, has been done. This has been due, not to a lack of the sulphide ore, blende, in the limestone, but to the fact that the soft or oxidized ores within the belt of weathering were adequate for the needs of the furnaces. Now that the known areas of soft ores seem very limited, considerable exploitation of the sulphide ore-bodies in the hard limestone is in progress over the Virginia-Tennessee region. In Virginia the outlook is very encouraging at Austinville and Cedar Springs in Wythe county, and at several points in Rye Valley in Smyth county. In Tennessee equally promising results are indicated in three areas: the Holston River area in Knox and Jefferson counties, the Straight Creek area in Claiborne county, and the Powell River area in Union county.

Preparation and Smelting of the Ores.

Washing and Milling.—The preparation and treatment of the ores can best be explained by describing the methods formerly used and at present practiced at the two largest and most extensively worked mines in the Virginia district—namely, Bertha and Austinville. At Bertha, mining for zinc-ores has been discontinued, and the mines are now being worked for brown iron-ore. However, a description of the former treatment of the Bertha zinc-ores will be of interest at this time.

The Bertha Ores.—The mining of zinc-ores at Bertha was stopped in 1898, when the property was leased to the Pulaski Iron Co., for the purpose of mining iron-ores. The hope exists, however, that more available zinc-ore will be found when the covering of iron-ore has been removed.

Two storage-bins for receiving the ore as mined were built on the bluff overlooking New river, about 0.75 mile from the mines. These were built on timbered trestles leading out from the hillside, and were provided with V-shaped floors, down the center of which passed a cast-iron water-trough, 12 in. wide and 6 in. deep. The ore was hauled in tram-cars from the mines to the bins, where it was fed into the water-trough and carried thence by a current of water to the dressing-house, 13 ft. below. The water used was pumped into a large tank above the bins on the hilltop, and there stored and fed as required. The in-

clined plane between the troughs was used to hoist, by steam, timbers and supplies for the mines.

The dressing-house was a three-story building, reported to have been well equipped with all the necessary automatic machinery for the concentration of zinc- and iron-ores. The zinc-ore was brought down the trough, falling upon a grizzly, through which the large lumps were broken. All the material was then passed into a single log-washer, where it was given a gentle sluicing, to free the ore from adhering clay.

Because of its porous structure and the clay contained in the cavities, it was necessary to treat the lump-ore further by passing it through a Blake breaker and a pair of Cornish rolls. From the rolls the ore passed to conical revolving screens, where it was sized; the large pieces dropped upon a steel-plate conveyor and were hand-picked. The small pieces passed down to four sets of Parson jigs, on which they were thoroughly concentrated. The tailings, resulting from this treatment, were passed through a spitzkasten or classifier, and thence to two Harz jigs. The slimes were discharged into a slime-pond, from which the muddy water was drained off into the river.

The capacity of the dressing-house is reported to have been 80 tons of concentrated zinc-ore per day of 10 hr. The yield is said to have been approximately one-third of the crude ore treated, and the product, dried at 212° F., is reported of the following average composition:—

ZnO, 47.61; (metallic Zn, 38.08); SiO_2, 29.37; Fe_2O_3, Al_2O_3, 9.23; $CaCO_3$, 4.54; $MgCO_3$, 2.07; H_2O (combined), 8.23; Pb, tr.; total, 101.05 per cent.

A roasting plant, consisting of an 8-ft. Taylor gas-producer and a 30-ft. cylindrical roaster, was operated for expelling the moisture from the ore.

The Austinville Ores.—The run-of-mine ore at Austinville is said to average in composition from 28 to 30 per cent. of zinc (metallic), from 8 to 10 per cent. of lead (metallic), and from 8 to 10 per cent. of iron (metallic). A brief sketch is here given of the method of treating the Austinville ore from the time it is mined until it is ready for the furnaces at Pulaski. A general view of the Austinville Zinc- and Lead-Mines plant is given in Fig. 3.

FIG. 2.—GENERAL VIEW IN THE MINE OPENINGS AT IVANHOE, VIRGINIA, SHOWING THE IRREGULAR LIMESTONE SURFACE WEATHERED INTO PINNACLES OR "CHIMNEYS." (Photograph by H. Ries)

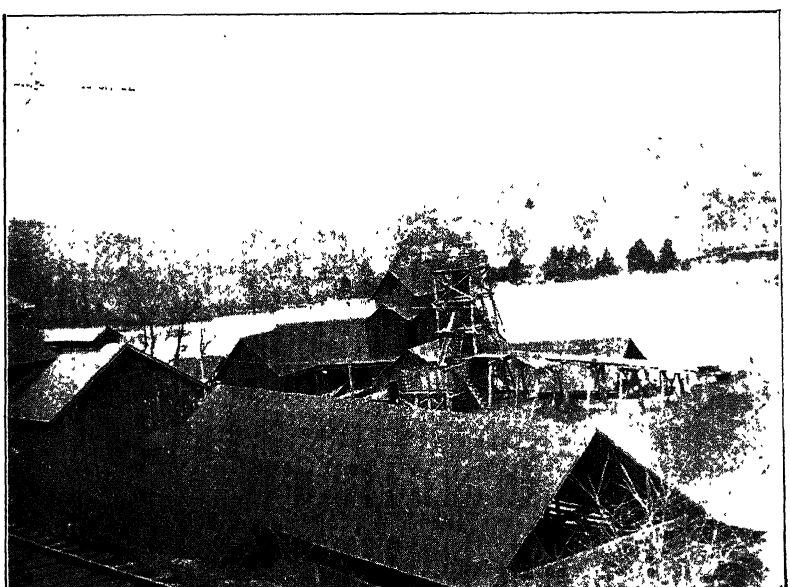

FIG. 3.—GENERAL VIEW OF THE AUSTINVILLE ZINC- AND LEAD-MINES PLANT.

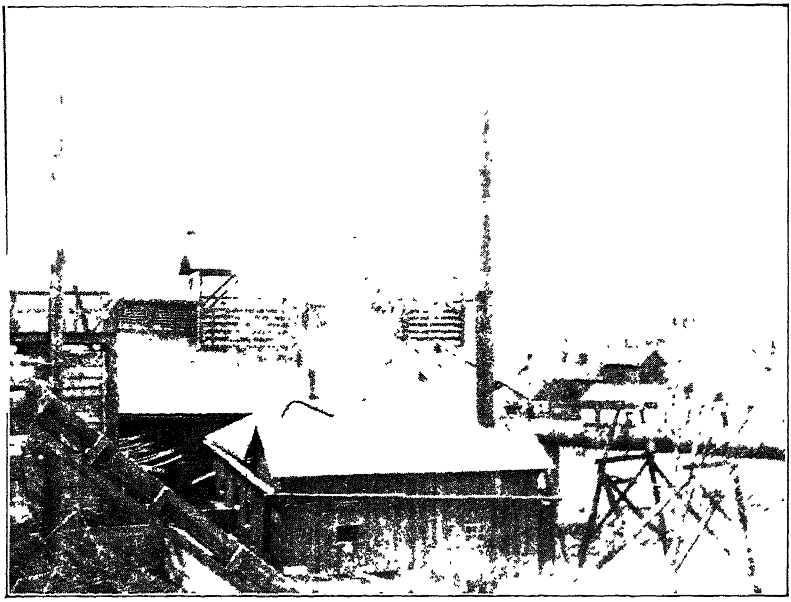

FIG. 4.—THE OXIDE-FURNACES AT AUSTINVILLE, VIRGINIA.

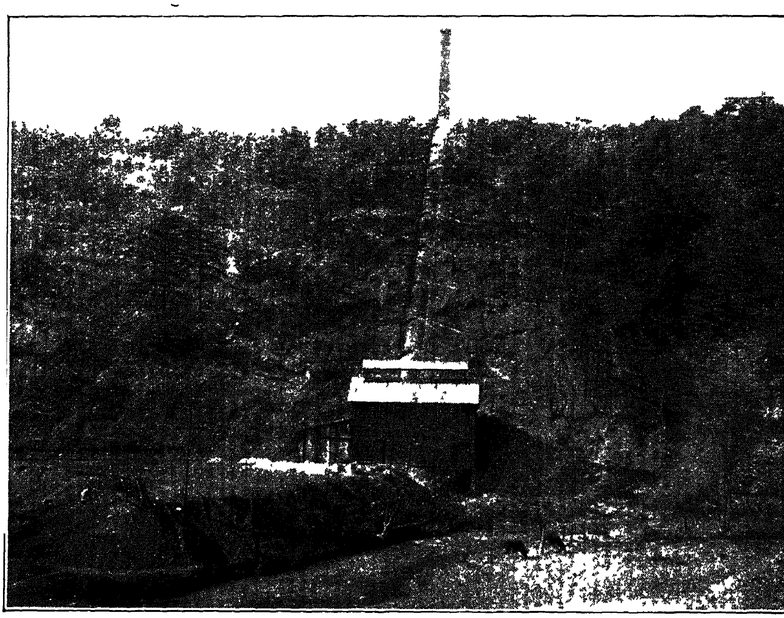

FIG. 5.—ROASTING-FURNACE FOR TREATING ZINC-SKIMMINGS.

The ore, as mined, is carried by trolley to tipples, from which it is carried by tram-cars to the mill, and dumped on a 4-in. grizzly; thence to a log-washer through a 9- by 15-in. Blake breaker set at 1.5 in.; and thence to 14- by 24-in. corrugated rolls set at 6 mm. From the corrugated rolls the ore is carried to a second log-washer, thence to elevator and sizing-screens of 6-mm., 2-mm., and 1-mm. mesh, respectively.

The oversize from the 6-mm. screen goes to smooth rolls, 12 by 18 in. in size, and is returned to the system or second log-washer. The undersize from the 6-mm. on 2-mm. screen goes to the 4-compartment Cooley jigs; 1st hutch and side-draw making clean lead; 2d hutch and side-draw, middling, re-crushed and returned to the system or second log-washer; 3d and 4th hutches, clean zinc. Tails are passed to settling-boxes; head of boxes reworked. Lighter material goes to the oxide furnaces.

Undersize from 2-mm. on 1-mm. screen goes to the 5-compartment jigs; 1st and 2d hutches and side-draw making clean lead; 3d hutch and side-draw, middling; 4th and 5th hutches and side-draw, clean zinc. Tails go to boxes.

Undersize from 1-mm. screen goes to a 3-compartment classifier; 1st and 2d draw to 6-compartment jigs; 1st and 2d hutches and side-draw making clean lead; 3d and 4th hutches and side-draw, middling; 5th and 6th hutches and side-draw, clean zinc. Tails go to boxes.

Third draw from classifier and overflow goes to the 3-compartment double jigs; 1st and 2d hutches making clean lead; 3d hutch, middling, reworked. Tails go to boxes.

The middlings and tails from all jigs are treated in 4-compartment jigs, as described above. The overflow from 1st and 2d log-washer, except the lower end or lighter material, goes to settling-boxes, 25 ft. long and 8 ft. wide; thence to the oxide furnace.

The lead-concentrates—averaging, lead, 57; zinc, 4.8; and iron, 3.29 per cent.—go to the Scotch hearth furnace, which makes metallic lead and gray slag, the latter averaging, lead, 31; zinc, 25; and iron, 13.3 per cent. This slag goes to the slag-furnace, which makes metallic lead and slag, averaging 3.5 per cent. of lead.

The zinc-concentrates are carried by tram-cars from the mill

to the separators, where they are put through a revolving dryer to expel moisture and combined water to the amount of 7 per cent. Thence to a water-cooled scraper-conveyor, to the top of building by elevator, to sizing-screens of 0.5-mm., 1-mm., 2-mm., and 0.25-in. holes. Oversize from 0.25-in. screen is returned to 12- by 14-in. rolls set close. Undersize re-enters the system. From screen to bins, four sizes.

The four sizes, which pass through screens, go to Wetherill magnetic separators, one 4- and one 6-magnet machine; the smallest sizes going to the smaller machine. Heads from separators, containing iron, 47, and zinc, from 5 to 7 per cent., are shipped to the iron-furnace. Tails from separators, containing zinc, 43.5; iron, from 5 to 6; and lead, 15 to 20 per cent., are shipped to the zinc-furnace. The capacity of the mill is about 90 tons of raw ore per day of 10 hours.

Zinc-Oxide Plant.—This plant, consisting of 12 furnaces, has only recently been added, mainly for the purpose of utilizing the low-grade ore and tailings, which were formerly discarded. After thorough testing, the plant has proved entirely satisfactory. The grade of ore, including tailings, used for making the oxide, will average from 15 to 20 per cent. of metallic zinc. The zinc oxide is conducted from the furnaces, through a 3-ft. iron cylinder, 600 ft. long, where it loses nearly all of its heat in transit, into bags. At present, the zinc oxide is sent to the Pulaski furnaces, where it is used for making spelter. No attempt is made to purify the zinc oxide sufficiently to market it as zinc-white. As now made, the zinc oxide is white, contains about 4 per cent. of lead, and averages from 70 to 80 per cent. of metallic zinc. A view of the zinc-oxide plant at Austinville, Va., is given in Fig. 4.

The successful operation of this plant at Austinville is of great importance, since it conclusively points out a method for utilizing low-grade zinc-ores, which elsewhere, it is claimed, cannot be profitably worked for making spelter.

The Bertha Smelting-Plant.

Location and Equipment.—The only zinc-smelting plant in the South, located at Pulaski, Va., is owned and operated by the Bertha Mineral Co. This plant was built soon after the beginning of zinc-mining at Bertha, Wythe county, Va., in 1879.

It was remodeled and enlarged in 1886, and, at present, it consists of 10 large smelting-furnaces, ore-sheds or bins, refiners, coal-pit, engine-room, pottery, metal-storage houses, and a roasting-furnace. Sufficient railway-trackage is in operation around the plant for shifting the cars; and a narrow-gauge road is operated between the plant and the company's coal-mine at Altoona, from which all the coal formerly used was derived; but, at present, only the "firing" coal or reducing-material is obtained.

Furnaces.—The 10 furnaces are of the Belgian type, hand-fired, built in 5 blocks of 2 furnaces each, placed back to back. The furnaces are 8 ft. deep, 30 ft. wide, and 24 ft. from top to bottom of foundation. They have an average capacity of 20,000 lb. of spelter every 24 hr. The capacity of each furnace varies from 1,800 to 3,000 lb. of metal per 24 hr., depending on the grade of ore used. Each furnace consists essentially of a large skeleton combustion-chamber, lined in the back with fire-brick, and having an iron frame-work front. The retorts are inserted from the front, with the back ends resting on a fire-brick support, built out from the back lining, so that when in position the retorts incline slightly toward the front of the furnace. Each furnace is fitted with 140 retorts, arranged in seven rows of 20 each.

The grates, placed from 3.5 to 4 ft. below the lowest row of retorts, are from 10 to 12 in. wide and 13 ft. long. A peculiarity is that the grates have, besides the three solid bars placed lengthwise, two hollow bars, arranged one on either side next to the wall. These are charged with water under pressure. Along the tops of the hollow bars are perforations, through which the water escapes when it is turned on at intervals. These water-sprays strike the under-side of the cinder, forming on the grate, and serve to break it up thoroughly, so that the ash is fine and pulverulent. This produces a clear fire and a cool ash-pit. Pocahontas coal is used, and, on account of its high grade and the clean grates resulting from the use of the water-spray, described above, its consumption is low. A fairly uniform heat is obtained over all the furnaces.

The fire-boxes are narrow, deep and long. The furnaces are fired from either end, the entire length of the grate-surface being 26 ft., with a sloping divide in the middle, and chimney-

draft. A very heavy bed is used, so that each furnace practically becomes a gas-producer. The doors are closed, and air is introduced through small holes in the front of the furnace, which can be closed as needed. Additional draft is provided above the furnace for regulating the heat.

Retorts and Condensers.—The retorts used are either round or oval. The dimensions of the round retorts are: length, 4 ft. 2 in.; inside diameter, 8 in.; thickness, 1.5 in.; capacity, 1.454 cu. ft. The oval or elliptical retorts, of a capacity of 1.228 cu. ft., are placed in the lower rows of the furnace, each retort being so placed that its greater diameter is in a vertical position. The retorts were all made at the company's plant on a Wooley tile-machine, of a capacity of 100 retorts per day of 10 hr. These retorts are made from the best grade of St. Louis, Mo., fire-clay, which is, perhaps, not so refractory as some other more siliceous clays, but is regarded as the best material for this purpose.

After coming from the machine, the retorts are arranged in a drying-room for two weeks, and held at a constant temperature of about 80° F. They are then placed in a second room, the temperature of which is 100° F., where they are kept for two months. Just before being placed in the furnace the retorts are stacked in kilns and fired for 16 hours.

The condensers are 21 in. long and 3.5 in. in diameter at one end, the other end being slightly smaller than the retorts. These are all made of local clay, mixed with powder made by grinding up old retorts. Shortly before being used or placed in position, they are washed inside with a mixture of water and fire-clay, in order to prevent the metal from soaking into them. The retorts are luted to the condensers with a mixture of one-third clay and two-thirds coal, enough water being added to bring the mixture to the proper consistency.

Materials Used in Charging.—At present, the materials used in charging are: local ores, including the silicate and carbonate of zinc; crude zinc oxide, containing from 70 to 80 per cent. of metallic zinc, and made from local ores at Austinville; willemite, anhydrous zinc silicate, from Franklin, N. J., averaging 48 per cent. of metallic zinc; zinc flue-dust from the iron-furnaces; and zinc-skimmings from the galvanizers, which are first given a preliminary roasting in a special furnace in order

to drive off the ammonia and chlorine. The zinc-skimmings are easily fired, and are usually smelted alone. No sulphide ores are handled at present. Fig. 5 is a view of the furnace for roasting zinc-skimmings.

As a rule, the ore, as delivered at the plant, is ready for the furnaces, it being sized to pass about a 6-mm. mesh screen. Some lump ore, however, is received, which is crushed and sized to a $\frac{1}{8}$-in. mesh size before using. The retort-charges are usually mixed, though some of the furnaces, as occasion demands, are run on special material alone, such as the zinc oxide from Austinville. The only difficulty in smelting the oxide alone is the slight tendency to "ball up," which makes it necessary to introduce a mechanical mixer in order to produce a more intimate contact between the ore and the carbon. For this reason the oxide is usually mixed with some ore before smelting.

The charges, consisting of ore and fuel, mixed in the proper proportions, are wheeled to the furnaces, moistened with water, and mixed on the hearth by shoveling. The charges are all mixed by weight. At present, the oxide, willemite, crude ore, coal and coke are mixed on the hearth, as described, and used for charging. The mixture and fuel are charged tightly into the retorts; a condenser is then inserted into the mouth of each retort, and luted and sealed perfectly tight.

Distillation and Tapping.—The furnaces having been made ready and fired, the reduction of the zinc in the retorts takes place according to the following formula: $ZnO + C = Zn + CO$. The CO acts as a further reducing agent, thus: $ZnO + CO = Zn + CO_2$. The zinc vapor passes from the retorts into the condensers, where it collects in the form of molten metallic zinc. Three taps are made during a run of 24 hr., and after the last tap is made, the fires are allowed to slacken, the condensers are removed, and the retorts are thoroughly cleaned with an iron rabble. Each retort is tested with a hook before recharging, so as to be sure of its being thoroughly clean; and it is also inspected for holes and cracks. It is claimed that the retorts cannot be cleaned by blowing out with steam, as practiced in the West, on account of the siliceous character of the materials used. The time required to complete the process at the Virginia furnaces is longer than at the western furnaces,

so that, in the case of the former, the furnaces are cooler for a longer time than at the western ones. The breakage of retorts is small, being only from 0.6 to 0.9 per ton of ore smelted.

Spelter.—Three grades of spelter are produced, branded, according to purity: "Bertha Pure Spelter;" "Old Dominion" and "Southern." The Bertha Pure Spelter has a world-wide reputation, and is sold under guarantee of 99.98 per cent. of metallic zinc. The following analyses afford an excellent idea of its purity:

	Sample I.	Sample II.	Sample III.
Zinc (by difference), ...	99 949	99.981	99.963
Iron, ...	0.010	0.019	0.012
Lead, ...	0.035	trace	0.012
Sulphur, ...	none	trace	0.001
Silicon, ...	0.206	none	0.202
Carbon, ...	trace	none	none
Arsenic, ...	none	none	none

As will be noted from the above analyses, the Bertha Pure Spelter is practically free from all impurities. It is extensively used abroad and in the United States for Government work. It is produced, in part, directly from the furnace; but it can be produced from either of the other brands, Old Dominion or Southern, by refining in a special patented furnace, in which the metal is redistilled carefully in order to avoid carrying over any of the lead. The Old Dominion brand of spelter contains from 0.2 to 0.4 per cent., and the Southern brand from 0.8 to 1 per cent. of lead.

The plant of the Bertha Mineral Co. is famed for the high grade of its spelter. To some extent this is due to the purity of the materials used, being specially free from iron, and to the fact that carbonate and silicate ores only have been used; the chief reason, however, is because of the good furnace-work.

Tests.

Some very interesting and important tests at the Pulaski plant have recently been conducted on low-grade mixed ores. One of the most important of these was on a lot of mixed carbonate and sulphide ores from north Arkansas. It was found necessary, first to give the mixture a roasting in order to desulphurize the blende present in the mixture. By giving the mixture a slow roasting the sulphur was driven off without loss

of the zinc in the carbonate, and, at the same time, the carbonate was benefited by having the carbon dioxide expelled. The success of this experiment has resulted in the willingness of the Bertha plant to treat the Arkansas ore, which, at present, is somewhat of a drug on the western market. This probably opens up a better future for the Arkansas ores, and, at the same time, points to an additional source of ore-supply for the Virginia plant.

Whenever it was possible to get Tennessee ore, that has been used. However, the lime present in the Tennessee sulphide ores seems to interfere with the work, since, by desulphurizing the ore, calcium sulphate is formed, which cannot be removed. Perhaps the principal reason for this is the finely disseminated character of this ore. This objection can be met, however, and a reasonably clean concentrate carrying little or no lime can be produced, by carefully milling, jigging and tabling the ore.

The Bertha Mineral Co. has not been producing ore from its property during the past year, for the reason that it has been and is still engaged in exploiting and developing the very promising bodies of sulphide ore at Austinville.

Future Development.

From the beginning of mining zinc-ores in southwestern Virginia to the present time, about 35 years, attention has been given exclusively to the removal of the soft or oxidized ores, the carbonate and the silicate. Now that the known supply of these ores in the district is rapidly diminishing, the problem in the future will be that of mining sulphide ores. Already considerable activity is manifested over both the Virginia and Tennessee districts in prospecting for and developing sulphide ores. Very encouraging results are obtained in places, and there seems sufficient reason for regarding the district as a promising one for workable bodies of sulphide ores.

The very favorable location of the district as regards abundant fuel-supplies, cheap labor and ample transportation-facilities, should make it possible to work lower-grade ores more profitably than in many of the central and western districts.

The ores yet found are in the Shenandoah limestone, or its equivalent in East Tennessee, the Knox dolomite. The Shen-

andoah limestone includes all of the Knox formation and at least 1,500 ft. of Cambrian strata beneath it. The ores apparently extend from near the bottom to the top of the limestone, and they are closely associated with the folding, faulting and brecciation of the district.

Tennessee.

Conditions similar to those in Virginia exist in the Tennessee district. The same ores are found, and, as in Virginia, much of the zinc-mining in Tennessee has been done on the soft ores, smithsonite and calamine, which are alteration-products from the original sulphide, sphalerite. At the present time, attention has been chiefly directed to the exploiting of sulphide ores, and, during the past 12 or 18 months, much testing has been in progress over parts of the district, principally in the form of drilling. This is particularly true of the Holston River area, in Knox and Jefferson counties, and the results seem quite encouraging. At least two other areas in Tennessee—namely, the Straight Creek and Powell River areas—show equally encouraging results in the presence of workable sulphide ores. From the very nature of the deposits, the methods of mining and preparation of the ores in the two States are closely similar. The Tennessee ores have been shipped, in part, to the Virginia smelter at Pulaski, and in part to the Indiana smelter at Marion.

A Novel Method of Mining Kaolin.

BY ALBERT R. LEDOUX, NEW YORK, N. Y.

(Bethlehem Meeting, February, 1906.)

I AM indebted to The Kaolin Co. of West Cornwall, Conn., and particularly to its engineer, Mr. M. Wanner, for permission to make public, through the *Transactions* of the American Institute of Mining Engineers, the interesting results of his experiments, having in view the winning of kaolin from deep deposits without shaft-sinking or removing of the overburden. So far as I am aware, nothing like it has been accomplished heretofore, although it reminds one of the methods successfully employed in Louisiana for the winning of sulphur from deep beds by the sinking of pipes one within another, injecting superheated steam under pressure through one pipe, the heat of the steam liquefying the sulphur and the pressure forcing it up through the other pipe to bins at the surface.

The kaolin-deposit at West Cornwall is an alteration *in situ*—that is, it is not sedimentary. A series of clay-veins, dipping about 50° from the vertical, lie between a foot-wall of limonite and a hanging-wall of gneiss and hornblende schist. The clay-veins alternate with veins or seams of more or less broken quartz, and unaltered feldspar. The deposit, which occurs at a point about 600 ft. above the Housatonic river, was opened five years ago, and about 5,000 tons of a superior grade of washed kaolin have been extracted from open pits, and sold. It soon became evident that this mode of extracting the product would prove unremunerative, by reason of the dip of the vein and the intercalary strata and lenses of quartz, and a more satisfactory method of working the deposit was sought. It is well understood that, in the preparation of all clays for market, especially residuary kaolin, the most important as well as the most difficult step is the washing of the crude product. The difficulties increase with the percentage of foreign matter, especially quartz and mica, contained in the product. Hence, a system

of working a clay-deposit by which it would be possible to obtain the crude material from great depths without removing the overburden (or, in the case of veins, without resorting to shafts, drifts and other underground workings requiring timbering), and, at the same time, minimize the raising of waste material with the marketable product, would be of great advantage.

Proceeding on the logic of these observations, Mr. Wanner, the engineer temporarily in charge of the operations at the mines, conceived the scheme of disintegrating the kaolin *in situ* by means of jets of water under sufficient pressure, and floating the resultant product to the surface. This product is technically known in the pottery trade as "slip." To accomplish this result, holes are drilled through the overlying gneiss, a pipe of 4-in. internal diameter is inserted into the bore and driven into the clay body to within a few feet of the foot-wall. The wells in operation are from 50 to 198 ft. deep. Into this 4-in. pipe, or "casing," an interior pipe of 2-in. external diameter is inserted, leaving an annular space of 1 in. for the upward flow of the slip. The lower end of the internal pipe is provided with a mouthpiece with several nozzle-like openings for the exit of the water; the mouthpiece rests on the clay body, and the interior pipe sinks gradually as the clay is removed until it rests on the foot-wall of the vein. For the operation of these "hydraulics" a head of water is required, equivalent to a pressure of from 40 to 60 lb. per sq. in., according to the nature of the vein-matter.

Residuary kaolin slacks more or less readily, according to the amount of sand and mica mixed with it. In the case in point, it has been found that a pressure of 40 lb. is amply sufficent to cause the disintegration. The vein-matter contains 20 per cent. and the slip, discharged by the hydraulics, from 60 to 75 per cent. of pure kaolin. The purity of the discharged slip is inversely proportional to the velocity of the overflow.

Observations made during the past summer's work have shown that the overflow contains from 5 to 10 per cent. of solid matter. A discharge of 100 gal. per min. through the annular space of 9.42 sq. in. from a depth of 127 ft. yielded 5 per cent. of solid matter, of which 75 per cent. was pure kaolin, while a discharge of 200 gal. per min. through the same orifice from

the same depth, gave a slip containing 10 per cent. of solid matter but only 54 per cent. of pure kaolin, the rest being finely-divided quartz and mica.

One of the factors directly responsible for the purity of the slip, as well as for the facility of the separation or washing of the product, is the accumulation of the waste material in the cavities formed by the withdrawal of the product, through which the slip has to pass to arrive at the lower orifice of the casing. This mass, which consists principally of a breccia of quartz, acts as a scouring medium, and the passage of the slip through it effectively helps in the separation of the clay particles from associated fine scales of mica, and renders easy its final division by subsidence.

In addition to the lessening of the cost of extraction, the method described has effectually solved the problem of transporting the product to the railroad. Heretofore, the kaolin, washed and dried at the mines, was carted by teams over a difficult mountain road to West Cornwall, 4 miles distant. The fuel for the whole plant had to be hauled up the mountain the same distance. With slip issuing from the hydraulics of only 10 per cent. of solid matter and sufficient fineness to pass through 100-mesh screens, the conveyance of the product through a pipe-line to the Housatonic valley offers no difficulty, and the company now contemplates the erection of a new washing-plant adjacent to the river and railroad.

Gold-Dredging in the Urals, with Notes on Dredging in Siberia.

BY WILLIAM H. SHOCKLEY, NEW YORK, N. Y.

(Bethlehem Meeting, February, 1906.)

[SECRETARY'S NOTE.—The following notes, arranged and edited in this office, but not yet revised by the author, were placed at my disposal with much modest hesitation (due to their incomplete and fragmentary character), upon my earnest request, and my argument that such recent and trustworthy data concerning a new and important mining industry ought not to be withheld from our *Transactions*, simply because they are not sufficient to constitute a complete academic treatise on the subject. This argument is recommended to the attention of many other members, who are waiting for the opportunity to contribute something "monumental" to our *Transactions*. In a number of instances, death has nullified this ambition, and the intended "monumental" contribution has turned out to be an obituary notice by the Secretary, instead of a valuable record of the professional experience thus hopelessly lost to the world.—R. W. R.]

The official publications of the Russian Government give fairly complete statistics of the dredges operated in the Empire; and in a recent English bluebook, Mr. Cook's report on Siberian Trade presents a brief account of dredging in Siberia. The data in Table I. (not given by Mr. Cook) have been taken from the *Gold Mining Messenger* (Russian) for July, 1905. These interesting statistics, though incomplete, show that the Yenisei dredges are working on poor ground and yield small returns. According to report, but few yield a profit exceeding 10 per cent. per annum on the capital invested.

In the summers of 1904 and 1905, I traveled in the Urals as far north as lat. 61° N., long. 61° E., on the Losva river, a part of the vast system of the Ob river. This clear-water stream starts from the summits of the Urals at an altitude of 4,000 ft., and flows through a flat country, densely wooded with larch, spruce, pine, fir, birch and poplar. The river is full of fish, and is navigable for good-sized steamers as far as Ivdell. Winter lasts from early November to late April, the temperature ranging from -60° F. to a summer maximum of 90° F. The rainfall is 20 in. The climate is healthy, although mosquitoes,

TABLE I.—*Data of Gold-Dredging in the Urals.*

	Size of Buckets, Cu Ft.	Speed Buckets, Per Min.	Nominal Capacity Per 24 hr.		Period of Operation, 1904.				Chief Cause of Shut-Down.	Total Quantity Worked.		Average Quantity Per 24 hr.		Value of Gravel.[e]		Production of Gold.[f]			Value of Gold Produced.	Total Expense.	Profit or Loss.
			Cubic[c] Sagenes	Cubic Yards.	Began.	Stopped.	Days Worked	Days Idle.		Cubic Sagenes.[c]	Cubic Yards.	Cubic[c] Sagenes	Cubic Yards	Doli[d] Per 100 Poods	Cents Per cu.yd	Poods	Zolotniks	Doli.	Rubles[g]	Rubles[g]	Rubles[g]
Dredges, Valley Yenisei River.[a]	4.5	11	90	[illegible]	Apr. 22	Oct. 25	157	30	Trommel	15,050	[illegible]	96	1,219.2	6.8	$0.1[illegible]	3	16	..	....	......	......
	5.0	11	100	1,270 0	Apr. 24	Oct. 27	144	12	Small	13,460	[illegible]	93	1,181 1	7.7	0.18[illegible]	3	14	56	....	....	......
	4.5	10	75	[illegible]	Apr. 29	Oct. 29	177	7	Small	17,092	[illegible]	96	1,219.2	3.3	[illegible]	1	33	50			
	4.5	10	85	[illegible]	Apr. 21	Oct. 29	187	4	Small	17,500	[illegible]	93	1,181 1	3.6	[illegible]	2	2	64	138,142	123,188	+14,954
	6.0	11	105	[illegible]	Apr. 25	Oct. 29	181	6	Tumblers	23,957	[illegible]	132	1,676.4	4.6	[illegible]	3	22	37			
	5.0	8	120	[illegible]	May 23	Oct. 7	132	6	....	6,227	[illegible]	47	596 9	7.5	0.180	1	21	52	27,800	......	......
	5.0	11	100	[illegible]	May 25	Oct. 29	147	10	....	10,175	[illegible]	69	876 3	7.0	[illegible]	2	13	13	42,500	33,500	+9,000
	5.0	11	120	[illegible]	May 9	Oct. 29	156	17	....	12,935	[illegible]	83	1,051.1	5.2	0.1[illegible]	2	8	45	40,565	50,765	—10,200
					Location of Dredge.																
Other Dredges[b]	3 0	20	110	1,397 0	Ivdell.		......	......	......	......	......	100	1,270 0	10 0	0.240	......	....	....	90,000	25,000	+65,000
	4.5	8	......	......	Bogoslovsk.		112	95	......	6,825	86,678	34	431 8	7.8	0 187	4	25	......	......	......	......
	4.5	8	......	......	Lobva River.		......	......	......	......	......	60	762 0	6.5	0.150	1	24	69	30,125	20,803	+ 9,322

[a] From the *Gold Mining Messenger* (Russian), July, 1905. [b] From personal notes of W. H. Shockley. [c] One cubic sagene equals 12.7 cubic yards. [d] One doli per 100 poods equals 2.4 c. per cubic yard (1 doli equals $\frac{1}{96}$ zolotnik). [e] Other dredges in the Yenisei region gave a yield of from 3 8 to 11.8 doli per 100 poods ($0.09 to $0.28). [f] A pood of pure gold is worth 21,156 rubles; a pood of alluvial gold in the Urals is worth 19,000 to 20,000 rubles, say $9,500. [g] One ruble equals 51.5 c. U. S. currency.

black flies and gnats are pests in summer. Winter is the season for travel, prospecting and forestry. The population, chiefly of Finnish type, is scanty and independent. Tartars, from Kazan on the Volga, do much of the hard work in the mines. There is but little agriculture, the people being supported by fishing, hunting and mining.

The route to this section is from St. Petersburg via the lately-finished railway to Viatka and Perm, to Goroblagodatskaia ("the blessed mountain"), thence by branch-railway to Bogoslovsk, and by post-horses to Ivdell; north of this, the travel is by boats in summer, and by reindeer-sledges in winter.

Ten dredges are now operated in this region, and eight more are under construction. It was thought that all would be operated in the summer of 1906; but political troubles have doubtless interfered with this programme.

The southernmost dredges, near Nijni Tura, belong to the Société Industrielle du Platine, of Paris, which employs 8,000 men, and practically controls the platinum-industry. There are two of these dredges, both formerly used on the Suez Canal. The first removes 12 ft. of top soil, which is not washed; and the second digs and washes the remaining 5 ft., which is pay-dirt. The expense of this double working is reported to be 25c. per cu. yd. Two Bucyrus dredges are being built for this company by the Putiloff works of St. Petersburg.

The San Galli Co. works a Bucyrus dredge in a swift stream flowing by slate cliffs, a few miles north of Nijni Tura. This stream, at first considered virgin ground, was afterwards found to have been worked in the past. The dredge excavates 1,200 cu. yd. per day, and, during the season, should take out 60,000 rubles' worth of gold at a cost of 30,000 rubles. The bed-rock is of hard slate; and boulders up to 300 lb. in weight are of common occurrence. Two iron pans, 15 ft. in diameter, with revolving arms, are provided to deal with the clay. While one pan is filled by the buckets, the outlet-valves of the other are closed, and the retained material is stirred until the clay is disintegrated and the rocks cleaned. In this manner, the dredging-operation is uninterrupted. A gate, shutting off the feed to the pans, and an iron chute, directing the dredged material directly to the tailings-belt, were provided; but, unfortunately, the water carried up by the buckets washed the dirt to the

lower end of the tailings-belt and prevented its working. In order to overcome this difficulty, it has been found necessary to treat the gravel in the pans, even though no clay be present.

Two dredges of the New Zealand type, made at Neviansk, near Ekaterinburg, are on the Lobva river near Bogoslovsk. One of these, with an iron pontoon, costing 90,000 rubles at the maker's works, when first put in the river, was hung up on a rock and nearly lost. The other, with a wooden pontoon, cost 50,000 rubles at the works. The total expenses, on account of the latter dredge, to August 1, 1905, are given in Table II.

TABLE II.—*Expense Account of Dredge at Lobva River.*

	Rubles.		Rubles.
General,	7,578.92	Wood,	609.02
Notary and legal,	1,400.80	Buildings,	4,144.68
Pontoon,	6,153.85	Materials,	6,672.94
Excavation for pontoon,	1,055.68	Payment for dredge,	50,100.00*
Prospecting,	1,395.20		84,461.59
Peasant proprietors,	343.36	Working expense for two months,	4,131.59
Salaries,	2,302.66		
Setting-up,	2,704.48		
		Total,	88,593.18

This dredge was built in the winter of 1904-5 in a pit excavated on the river-bank. During the construction, chips, shavings, hay, and manure from the teams employed in hauling the dirt, covered the ice in the pit, and, by protecting it from the sun's rays, retarded the spring thawing a full month. During this delay, the river fell 10 ft. lower than the dredge. Instead of digging a diagonal canal to the river, the pit was deepened vertically and the tailings were removed by teams. Another month was occupied in this work, giving a total loss of two months, due to bad management.

At Bogoslovsk, 1,663 prospecting-shafts, of an average depth of 1.75 sagenes (12.25 ft.), cost, on the average, 17.03 rubles ($8.50), equivalent to $0.70 per foot. In Alaska, similar shafts cost from $3.50 to $8 per foot.

Neither of the two dredges on the Lobva river yielded any profit during 1905, although the gravel carried 16c. per cu. yard. In 1904, the Bogoslovsk estate began work with a Neviansk dredge of the New Zealand type on the bed and banks of a small sluggish stream, having a total width of 200 ft. The gravel is soft for a depth of 14 ft., with 5 ft. of pay-dirt. This

company is building two similar dredges of the New Zealand type, somewhat improved by a study of illustrated catalogues and of the Bucyrus dredges working in the region. A Neviansk dredge on the Sosva river, 60 miles north of Bogoslovsk, stuck on a rock at the beginning of the season, and in three months produced only 1.5 lb. of gold.

The most successful dredging in the Urals is at Ivdell, a pretty hamlet 90 miles north of Bogoslovsk; a photographic view of this town is shown in Fig. 1. A Bucyrus dredge, built by the Putiloff works under the supervision of H. L. Lawson, an American dredge-master, was operated almost without a stop during its first season in 1905. Fig. 2 is a photograph of this dredge, which shows the steep banks of the Ivdell river in the background. It has dredged nearly a mile of the Ivdell river, a swift stream 200 ft. wide, flowing through a limestone formation. The amount excavated daily is estimated at 1,500 cu. yards. The cost of the dredge complete, including a Keystone drill and some prospecting-work, was 140,000 rubles; and the profit for 1905 is estimated at 65,000 rubles. A duplicate dredge, estimated to cost 100,000 rubles, is under construction. On all these dredges, tables are used to save the gold; the Ivdell dredge has shaking screens; the others, revolving trommels.

Figs. 3 and 4 illustrate the method of dredging by hand on the Ivdell river, September, 1905.

TABLE III.—*Cost of Operating the Dredge at Ivdell, in Rubles, Worth Each About* 50c. *U. S. Money.*

	Per Month.	Total.
3 pilots,	@ 75	225
3 machinists,	@ 75	225
3 firemen,	@ 30	90
3 oilers,	@ 40	120
2 woodmen,	@ 35	70
2 clean-up men,	@ 25	50
1 blacksmith,	@ 40	40
1 gold-washer,	@ 30	30
1 dredge-master (American),	@ 360	360
Wood,	@ 300	300
	Total,	1,510

The maximum expense per month, including repairs and all materials used, is estimated at 3,000 rubles, or $1,500.

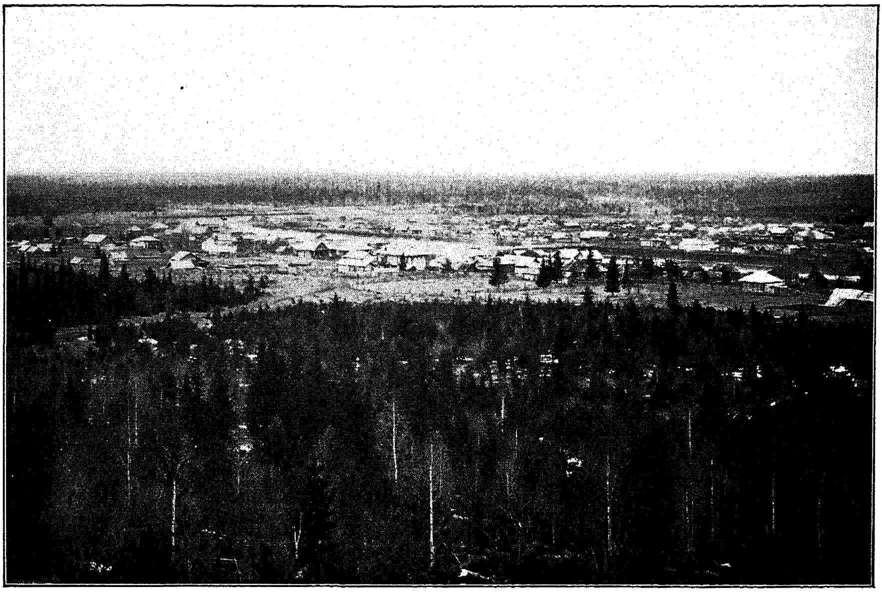

FIG. 1.—TOWN OF IVDELL, NORTHERN URALS.

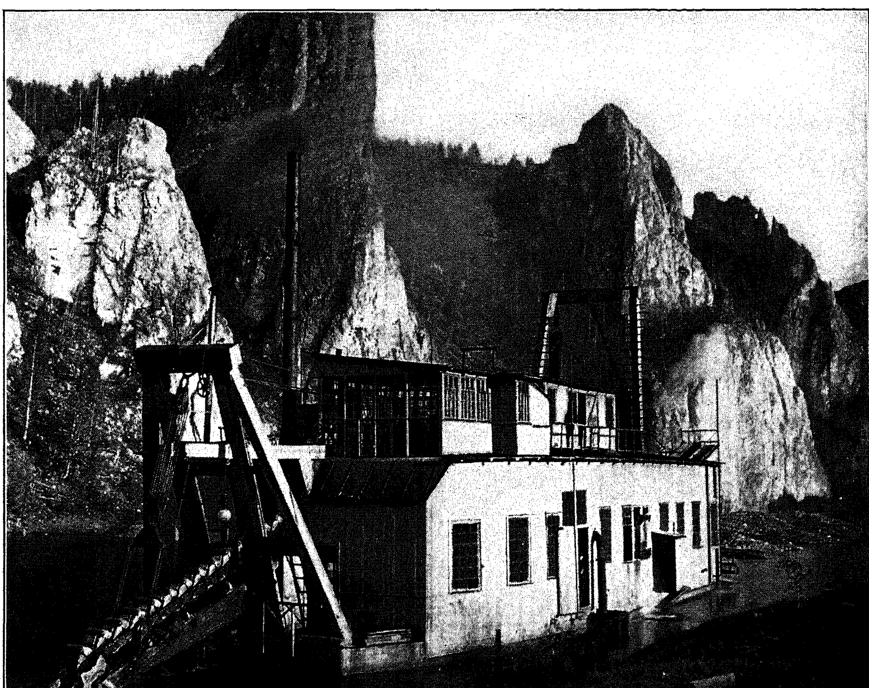

FIG. 2.—"BUCYRUS" DREDGE ON IVDELL RIVER, AUGUST, 1905.

FIG. 3.—"STARATELI" HAND-DREDGING ON IVDELL RIVER, SEPTEMBER, 1905.

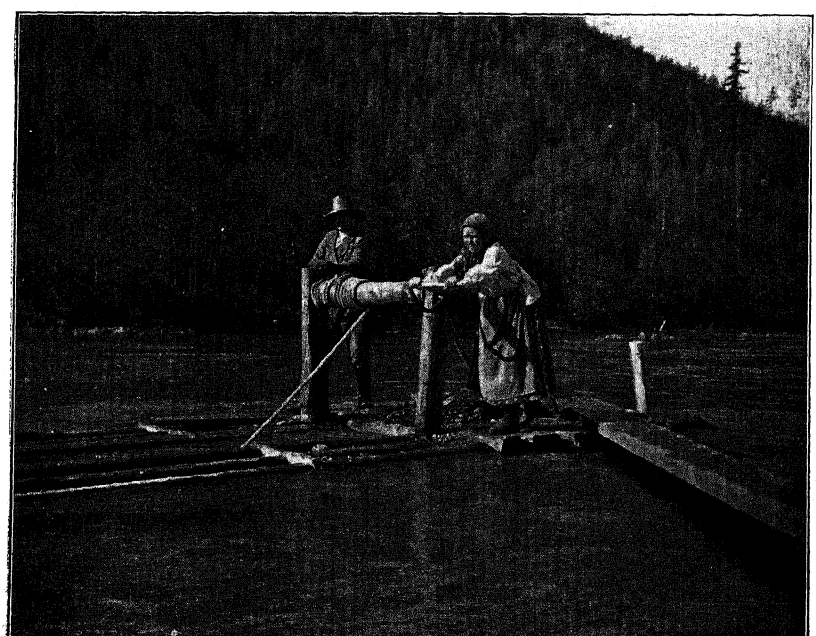

FIG. 4.—HAND-DREDGING ON IVDELL RIVER, SEPTEMBER, 1905.

Table IV.—*Cost of Supplies at Ivdell.*

	United States Currency.		United States Currency.
Butter, per lb.,	$0.20	Meat, moose, per lb.,	$0.03
Fish, fresh, per lb.,	0.06	Meat, reindeer, per lb.,	0.05
Fish, salt, per lb.,	0.03	Potatoes, per lb.,	0.004
Flour, per lb.,	0.013	Sable skins, each,	7.50
Hay, per ton,	5.50	Sugar, per lb.,	0.10
Labor, ordinary, per day,	$0.30 to .50	Wood, per cord,	0.50
Lumber, per 1,000 ft.	7.50	Reindeer, alive, per head,	7.50
Meat, beef, per lb.,	0.05		

From my observations, and from conversations with H. L. Lawson, dredge-master at Ivdell—an American with 10 years' experience in Montana and Idaho—I conclude that, for successful dredging in the Urals, the following conditions should be assured:

1. The work should be under charge of an American or New Zealand dredge-master, who has had experience in cold countries. This is a vital point. The Russian engineers, though well-educated and intelligent, are poor managers.

2. Extensive prospecting is needed, and can best be done in winter, by sinking shafts on the Siberian system—*i. e.*, allowing the water to freeze, sinking a short distance, waiting for the water to freeze again, and eventually reaching the bed of the river, even through running water. (This method, however, sometimes fails in the Urals, owing to unfavorable weather, or the presence of warm springs.) Hand-dredges, worked by parties of six men (or often by three men and three women), and washing a few yards daily, are useful for prospecting river-beds. A Keystone drill should be used in the river-banks by every dredging-company. In most of the Ural rivers, the chief values seem to be in the streams, the banks paying very little.

3. A small dredge, run by steam or horse-power, and costing, complete, not more than $5,000, would be of value on these rivers.

4. Dredges should be built on the bank and launched, in order to avoid the expense of a pit. Wooden pontoons should be used.

5. If prospecting-work shows that there is much clay in the gravel, the enterprise had best be abandoned, because no

method of dredging has yet been devised that will handle successfully material of this character.

6. The gold, which is uniformly coarse, should be saved in iron-lined, wooden sluices, and not on tables. These sluices should be 120 ft. long, like some used in Montana dredging. The question of sluices versus tables is still in dispute; but the advantages of the former seem to me decisive. Nuggets which pass over ordinary tables can be saved in sluices. Repairs to sluices are trifling compared with the expense of keeping up the belts of tables. Moreover, in a cold climate, a sluice can be operated for a number of days longer than a belt or a bucket-elevator. The water flowing in a large stream does not freeze so readily; and hence it is not necessary to shut down when the first cold snap comes.

7. Grizzlies should be provided to remove the large stones, which should first be washed.

8. Two boilers are needed, steam being kept in one while the other is being cleaned. In this way less time is lost; and the extra boiler is also useful for other purposes. All driving-shafts should be bored longitudinally, so as to allow an obstinate wheel or gear to be loosened for removal by heating, the shaft being kept cool by a stream of water flowing through it.

9. For swift rivers, buckets of 3 cu. ft. capacity are large enough. These should be closely set, and should excavate from 1,500 to 2,000 cu. yd. daily. For a dredge of this capacity, the following engines are advised: 60-h.p. for digging; 60-h.p. for 14-in. pump; 15-h.p. for swinging; 15-h.p. for trommel; 15-h.p. for elevating tailings (not needed if sluice is used); and 20-h.p. for electric light.

10. For swift streams, spuds are preferable to head-lines for holding the dredge in place. The top tumbler-shaft should be adjustable.

The mineral resources of the Urals are very great, and there is an immense field for dredging in the Russian Empire. When the present political troubles have passed away, the industry will exhibit a rapid development.

The Russian laborer is very good, considering his wage; and the officials, though sometimes troublesome, yield to tact and other influences.

Crushing-Tests of the Diamonds Used in Drilling.

BY PROF. ALEXANDER N. MITINSKY, MINING ACADEMY OF EMPRESS CATHERINE II., ST. PETERSBURG, RUSSIA.

(Bethlehem Meeting, February, 1906.)

UP to a certain limit, the increase of pressure on the diamond-drill increases the rate of progress in drilling. That limit is set by the resistance of the diamonds to compression; and beyond it, the destruction of the stones will involve an economic loss, both in time and in money. It is, therefore, important to all diamond-drillers to know what is the resistance mentioned.

In Europe, operators usually follow an empirical rule, which directs the maintenance of a pressure, on the area of the bottom of the drilling-tool, of 12 kg. per sq. cm. For the tools in ordinary use, this is equivalent to 1 kg. per sq. mm. of the diamonds set in the tool.

So far as I know, American drillers are using pressures as high as 50 to 60 kg. per sq. cm. of the total area of the drill, and the consumption of diamonds (on Denny's authority, 0.2 carat per meter bored) is greater in America than in Europe. But the reason for this difference may be the comparatively larger size of the diamonds commonly used by American drillers.

With the assistance of my friend, the late S. Woisslaw, who was the pioneer of diamond-drilling in Russia, I selected from his stock a number of black diamonds, which I subjected in my laboratory to compression-tests.

To prepare regular cubes for such tests, as is generally done with other stones, would have been not only costly, but, what is more important, practically useless. I therefore determined, in order to assimilate as nearly as possible the conditions of actual use, to place each sample between a hard plate (corresponding to the rock) and a softer plate (corresponding to the bottom of the tool), into which it would be pressed during the test.

In my first experiment, I placed a diamond between a plate of rolled steel and one of soft bronze, under a very sensitive press having a diagrammatic recording-apparatus, with a maximum load of 1 metric ton. The diamond endured the pressure of 1 ton, and penetrated into both plates without breaking. The area of its impress in the plates could not be determined with any degree of accuracy by ordinary methods; but by taking photographs, enlarging them, and then measuring them with a planimeter, I found the area of the impression in the bronze plate to be 19.5, and in the steel plate, 17.25 sq. mm. The half-sum of these areas, 18.4 sq. mm., being taken as the area of the diamond submitted to pressure, indicates that, under the pressure of 1 ton, exerted by the press, the diamond resisted 54.3 kg. per sq. mm. without breaking.

In my second experiment, I put a diamond (a very small one) between hard tool-steel and rolled soft steel. It broke under 370 kg., leaving on the mild steel an impression of 9.1, and on the tool-steel one of 3.3 sq. mm., corresponding to breaking-pressure per sq. mm. of 45.5 and 115.6 kg., respectively, or an average strength of 80.6 kg.

Subsequently, I compressed three diamonds successively between two pieces of steel of the tensile strength of from 40 to 50 kg. per sq. mm. After these tests, the diamonds were so firmly held in the plates that it was difficult to remove them. The difference of areas of the impressions in the two plates amounted to 8 per cent. The results of the test were:

Experiment No.	Total Load. Kg	Half-sum of Areas. Sq. mm.	Breaking Strength. Kg. per sq. mm.
3	620	8.0	77.5
4	993	17.5	56.7
5	768	11.0	70.0
		Average, . .	68.0

In view of these tests, I think that the pressure on drilling-tools can safely be made heavier, with corresponding gain in speed of drilling. American operators are already in advance of Europeans, but could go even further in this direction.

In Russia, the makers of diamond-tools frequently use a patented method of inserting the stones, which consists in enveloping the diamond in a piece of thin steel; putting it, together

with some welding-powder, under pressure in the tool, which has been heated to incandescence; hammering, and then cooling very quickly.

After my tests had been made, Mr. Woisslaw ordered his workmen to leave the danger of breakage out of consideration, and to put upon the diamonds the highest pressure attainable with their machinery. The results were very good. By this method tools can be obtained which will run safely at a peripheral velocity of 25 m. per second.

Notes on the Roumanian Oil-Fields.

BY P. CHARTERIS A. STEWART, CAIRO, EGYPT.

(Bethlehem Meeting, February, 1906.)

THE following scanty notes on the Roumanian oil-region may serve as an introduction to more detailed future study and description.

The Roumanian oil-belt follows the outer edge of the sweep of the Carpathian mountains, and may, in a broad sense, be regarded as a prolongation of the Galician belt. It is distributed over a tract of country from 300 to 400 miles in length, with a width of from 15 to 20 miles, and is believed to cover an area of at least 20,000 hectares (49,420 acres). The government claims that 16,000 to 17,000 hectares of its land is petroliferous.

The primary deposits of petroleum are considered to be in the Paleogene (Oligocene) and the Neogene (the salines of the Miocene) formations. Most of the other repositories, especially those of Muntenie, are only secondary, the petroleum there having been introduced by the orogenic movements which raised the Carpathian mountains.

The oil-field has been divided by the Roumanian Survey into two regions, the Flysch and the Sub-Carpathian, as is shown in the accompanying sketch-map, Fig. 1. So far as is yet known, practically all the seepages and productive pits and wells are situated on anticlinals.

Those which outcrop in the *Flysch region* (called Paleogene anticlinals), as indicated in the sketch-map, Fig. 1, are: *Solontz*,

Middle Oligocene; *Moineshti*, Moineshti and Tagu-Ocna beds of the Eocene.

Those of the *Sub-Carpathian* are:

Prajol-Campeni, Saliferous Miocene;

Beciu-Bercea, Meotic, and a few Pontic beds;

Sarata Monteor, Nucleus of Sarmation with Pliocene on the flanks. Direction of anticlinal, SW. byW.;

Recea, Meotic, with a kernel of massive rock-salt;

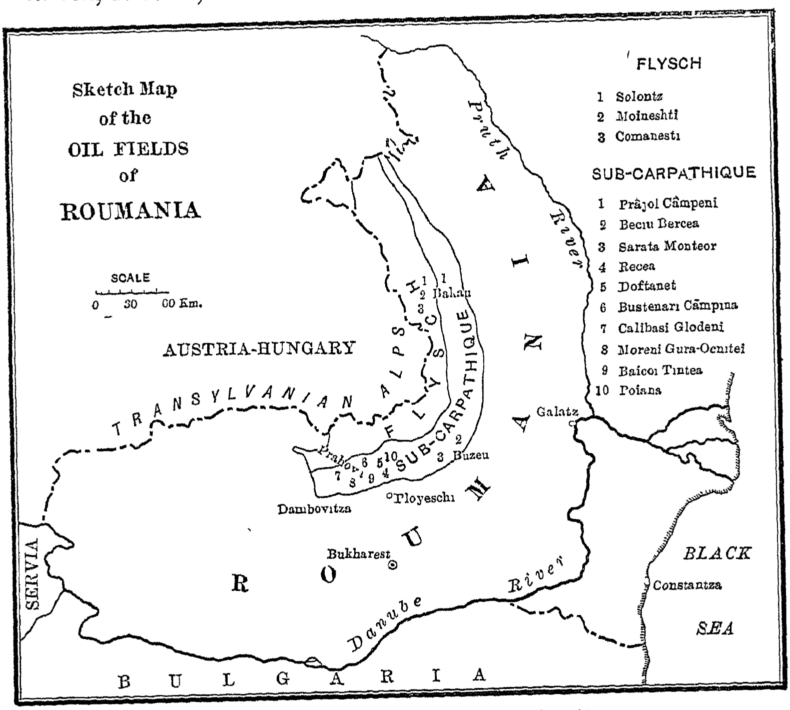

FIG. 1.—SKETCH-MAP OF ROUMANIAN OIL-FIELDS.

Campina-Bustenari-Poiana, Meotic, with rock-salt masses in places. (The eastern end of Bustenari *Faget* is Oligocene.);

Calibasi, Nucleus of Saliferous Miocene with Sarmatian, Meotic, Pontic (Congeria beds), and in some parts Levantine (Unio Sculptée beds);

Baicoi-Tintea, Anticlinal of Pontic (Vivipara bifarcinata and Congeries beds), and Levantine, lying directly upon the Saliferous Miocene;

Moreni, *Gura-Ocnitei*, Anticlinal of Pontic (Congeries and Vivapara); Levantine (Unios sculptées and Candeshti beds), lying directly upon the Saline Miocene.

The oil of the Miocene is found on the sandy-marl *facies* in the immediate vicinity of the salt masses. The general impression gained is that large quantities of petroleum may be found accumulated in the anticlinals which have a kernel of massive salt.

Most of the anticlinals of the Sub-Carpathian region belong to the Neogene epoch. They run more or less parallel with the main mountain-chain. Thus in the north, in Moldavia, their course is roughly N–S., while in the Prahova district it is more nearly ENE–WSW. By reason of the permeability of the grits and sands, the petroleum is generally contained in these rocks.

Erosion has, in some cases, carved deeply into the crowns of these anticlinals, laying bare the oil-strata, allowing the escape of oil and the ingress of water, with consequent flooding of the strata. In many places, the former impermeable covering is no longer found intact.

In the Flysch of Moldavia, some beds of which are supposed to be synchronous with those of Galicia, the relative absence of confined permeable beds seems to be the cause of a small production, as compared with that of Galicia. It is, however, possible that, more favorable conditions permitting, better results may be encountered in depth.

Numerous difficulties attend the drilling of wells in Roumania, the structure being, in many cases, very complicated. On the other hand, the Roumanians are inclined, in complaining of their own troubles, to underestimate the complexity of the strata and the difficulties of drilling in other parts of the world. The main obstacles with which Roumanians have to contend in drilling are: water, which must be shut out, and the breaking-off (after the removal of the oil-sand by the flow of oil and gas) of a hard, highly-inclined layer immediately above the oil-sand, which causes squeezing of the casing, and a caving of argillaceous bands directly overlying the oil-sand. Where the oil is found in overturned anticlinals, there is often a slipping along lamination-planes, which results in enormous pressures on the casing. Moreover, the occurrence of quicksands and of movements due to the removal of sand by the flowing gas and oil, may be the causes of disaster. To all these sources may be attributed many failures in the "getting down" of Roumanian wells.

There are about 87 localities where petroleum is known to exist; and of these, only about half-a-dozen have been anything like sufficiently exploited.

The figures in Tables I., II. and III. show that the Prahova district, with its production of 92.4 per cent., of which 82.3 per cent. comes from the two fields of Bustenari and Campina-Poiana, is by far the most important district in Roumania at the present time.

TABLE I.—*Production of Petroleum in Roumania from* 1895 *to* 1904 (*not Including the Quantity Used Locally for Fuel*).[a]

Year.	Metric Tons.	Year.	Metric Tons.
1895,	240	1901,	270,000
1896,	....	1902,	320,000
1897,	110,000	1903,	[b]384,302
1898,	180,000	1904,	500,561
1899,	250,000	1905,	614,870
1900,	250,000		

[a] Reported by the *Moniteur de Pétrole Roumain.*

[b] The return given by the Special Commission for 1903 was 388,090 tons, probably including the oil used for fuel on the field.

TABLE II.—*Production of Petroleum in Roumania for* 1902, '03, *and* '04, *Classified by Fields* (*Metric Tons*).

	Prahova District.					Dambo-vitza.	Buzeu.	Bacau.
1902	270,000					15,000	4,000	11,000
1903	345,913					22,469	5,920	....
	Bustenari.	Campina-Poiana.	Moreni.	Tintea.	Other Prahova Fields			
1904	331,860	109,269	4,349	4,100	5,776	26,234	8,828	10,145
1905	411,407	94,860	47,243	7,412	7,371	24,703	12,904	8,974
	(66.9%)	(15.4%)	(7.7%)	(1.2%)	(1.2%)			
	(92.4%)					(4.0%)	(2.1%)	(1.5%)

TABLE III.—*Data of Wells and Pits During* 1905.[a]

	Wells.			Pits.		
	Producing.	Drilling.	Abandoned.	Producing.	Sinking.	Abandoned.
Prahova..........	266	197	142	276	70	723
Dambovitza......	10	13	10	93	21	145
Buzeu..............	8	....	28	67	16	168
Bacau..............	46	5	42	244	26	194
1905..............	330	215	222	680	133	1,132 (?)
1904..............	220	107	148	743	175	1,129
1903..............	190	92	118	675	163	997

[a] Reported by the *Moniteur de Pétrole Roumain.*

Campina Bustenari.—This zone, about 14 km. long, with undetermined but varying width, is formed of an anticlinal of Meotic beds jammed up against a fault which is the southern limit of the Sub-Carpathian saline series. A section of this zone is shown in Fig. 2.

The eastern part of this belt, Doftanetz Faget, is formed of an anticlinal in Oligocene beds. A section of this zone is shown in Fig. 3.

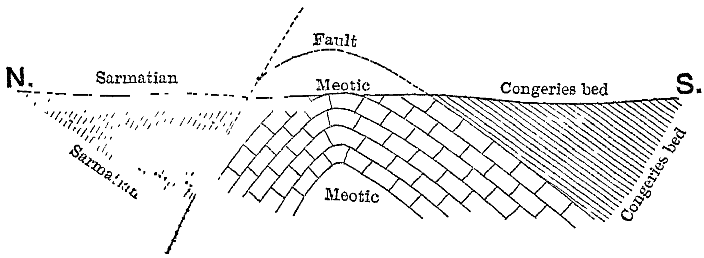

FIG. 2.—SECTION ACROSS OIL-ZONE AT CAMPINA.

Campina.—The strike of the anticlinal is N. 60° E., the dip to S. is about 30°, that to N. nearly 70°. There are about seven recognized oil-sands on the south side, but they are not all of a paying quality. The northern limb is steep and short, being cut off by the before-mentioned fault, causing the northern side to be of comparatively little value. This fault can be well seen in the Telega valley.

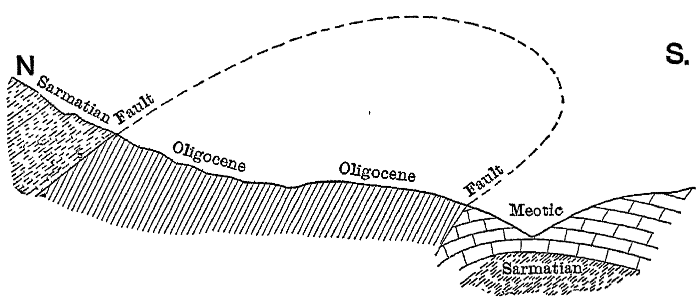

FIG. 3.—SECTION ACROSS OIL-ZONE AT FAGET.

Methods of Production.—Roumania has been, *par excellence*, the country of the "dug pit," and it is the pioneer work done by the peasants and others in proving the shallow beds that has encouraged the capitalist to come with the drill. As the peasants and smaller workers have been bought out, in order to drill for the deep oil, they have moved on to explore new lands.

These "hand-dugs" are surprisingly deep, at times attaining 750 ft. The cost of a hand-dug of 650 ft., with its lining of impermeable clay and wood-work, together with the requisite tanks, is given as 17,500 francs. These hand-dugs often give a steady, though small, yield for periods ranging up to 20 years. Flooding, "gassing" of the workers, explosions and quicksands are their chief obstacles. But good average hand-dugs will give a clear profit of 2,000 to 2,200 francs per annum, besides repaying their original cost.

Drilled Wells.—These are coming rapidly into vogue, but the deepest ones (*i.e.*, Campina, 2,600 ft.) have not been producers, the deepest producing wells being usually from 1,600 to 2,000, while the shallowest are only 130 ft. deep.

A good average well should yield about 30,000 kg. of petroleum per day.

. POSTSCRIPT.*—For the better understanding of my paper on Roumanian Oil-Fields, it would be well if the following list of Roumanian formations were added, since the terms used in the text are more or less peculiar to Southeastern Europe.

TABLE IV.—*Rocks Connected with Oil-Bearing Formations.*

Neogene.	Levantine.	Candeshti beds.
Neogene.	Levantine.	Unios sculptées beds.
Neogene.	Pontic.	Vivipara bifarcinata beds.
Neogene.	Pontic.	Congeries beds.
Neogene.		Meotic.
Neogene.		Sarmatian.
Neogene.		Saline Sub-Carpathian.
Paleogene.	Oligocene.	Kliva Grits.
Paleogene.	Oligocene.	Menilite Schists (Shipota beds).
Paleogene.	Eocene.	Targu Ocna beds.
Paleogene.	Eocene.	Saline Paleogene.
Paleogene.	Eocene.	Eocene.

* Received July 20, 1906.

Biographical Notice of George H. Eldridge.

BY S. F. EMMONS, WASHINGTON, D.C.

(Bethlehem Meeting, February, 1906.)

By far the greater number of the members of this Institute are men who are engaged in the strenuous work of the technical part of their profession, and find little time for the abstract scientific work which is less distinctly practical and pecuniarily remunerative. There is, however, a small but increasingly numerous class whose members devote themselves to broad geological problems which underlie the systematic development, and whose work, theoretically, ought to precede and form a basis for that of their more practical brethren. These are the so-called mining or economic geologists, who are, for the most part, connected with State or National geological surveys. From the character of their work, the life of these men necessarily follows rather different lines from that of most of their colleagues, and is not generally familiar to them. It seems admissible, therefore, that to the life of a prominent member of this class, who died during the past year, having literally sacrified his life to his work, a more extended notice should be given than the relatively small number of his contributions to the *Transactions* might seem to justify.

George Homans Eldridge, son of Ellery and Sarah (Mathews) Eldridge, was born on Cape Cod at Yarmouth, Mass., Christmas day, 1854. His boyhood education in the public school at Yarmouth was followed by a six years' course in the Public Latin School at Boston, from which he entered Harvard University in 1872, being graduated with the highest rank in his class in the natural sciences in 1876.

During his college course he spent the summers of 1875–6 at Cumberland Gap, Ky., as volunteer aid to Prof. N. S. Shaler, who was then in charge of the State Geological Survey of Kentucky. This vacation work, undertaken by him rather as a pastime, gave him his first leaning toward geology. Soon

after graduation, through the failure of one of the trustees, his father's estate became so impaired that he found himself obliged to give up his chosen profession of medicine, and to embrace the more immediately remunerative one of teaching, in order to meet the responsibilities thus early thrust upon him. To this work he devoted himself with such assiduity and success that he was early appointed principal of the high school at Nahant, Mass. His was, however, too energetic a nature to allow itself to be tied down to the routine of a simple pedagogue. He organized a series of lectures on the natural sciences for the little community in which he lived, and still found time to take private lessons on the practical side of geological work from his former inspiring teacher, Prof. Shaler of Harvard.

It was early in 1879, while still in charge of this school, that the opportunity presented itself which ultimately led to the adoption of his life work, that of economic geologist on the United States Geological Survey. It was in this year that that institution was created as a permanent branch of the government service to succeed all previous and temporary organizations of its kind. During the same year, Gen. F. A. Walker had been placed in charge of the Tenth Census of the United States, with unusually wide powers and large appropriations. It was his broad conception that the census work should be something more than a mere compilation of dry figures. He thought it should also include such a critical study of the principal branches of industry of the country as would serve to guide Congress in its economic legislation during the following decade. To accomplish this, he placed the investigation of the different industries in charge of the men best fitted for the respective work, under whom the personal investigations should be made by experts chosen for their special qualifications. In pursuance of this plan, the Director of the new Geological Survey was given charge of the investigation of the precious metals, and to Prof. Raphael Pumpelly was intrusted that of the other useful metals, principal among which, at that time, were iron and steel.

Among the first special experts engaged by Prof. Pumpelly for the active part of this work was Eldridge, whose fitness for this new and peculiarly difficult work had been testified to by

Prof. Shaler, from his intimate knowledge of the qualities of the man gained during the vacation work under him in Kentucky. Special branches of investigation to which he was assigned were the base-metal deposits of the southern Appalachians and the coal-fields of northern Montana, which, at that time, were practically unexplored.

Two years later Prof. Pumpelly took charge of the Northern Transcontinental Survey, which had been organized by Henry Villard, then President of the Northern Pacific railroad, for the purpose of making a study of the mineral resources of the region traversed by that road. Eldridge was one of the first geologists employed by him, and was given charge of the Rocky Mountain division, to which was intrusted the solution of some of the most difficult and critical problems involved. Some of the results of this work are found in the *Reports of the Tenth Census* (volume xiii.); but, owing to its sudden termination, the results of the Northern Transcontinental Survey were never fully published.

When, in the winter of 1883–4, financial disaster overtook the Northern Pacific railroad, and the Northern Transcontinental Survey was, in consequence, suddenly disbanded, of the geologists thus set free, Eldridge, on the recommendation of his former chief, Prof. Pumpelly, was the first to be transferred to the United States Geological Survey, to fill a vacancy then existing in its corps of economic geologists.

From 1884 to 1890 Eldridge was attached to the division of mining geology, under my direction, and his work during this time lay mainly in Colorado. Eldridge early distinguished himself for his keen insight into stratigraphic problems, especially those connected with the underground development of coal-seams. Among his fellows his most striking characteristics were his untiring energy and the thoroughness with which he mastered every detail of his subject. His physical endurance was phenomenal. In camp-life in the mountains, where, at that time, the physical difficulties to be overcome were greater than at the present day, he seemed to gather new energy from every obstacle that came in his way; and while always cheerful, he became eminently so where others, through an accumulation of difficulties, were inclined to be discouraged. Winter work in these elevated regions had hitherto been con-

sidered inadvisable, but Eldridge demonstrated for the first time that such work was practicable by remaining in camp sometimes through the entire winter. During these years he was mainly occupied with stratigraphic problems connected with the southern Elk mountains in Gunnison county, Colo., and with the Denver basin on the eastern foot-hills of the Rocky mountains. In each case the completion of the work was unavoidably delayed by the inability of the topographic force of the Survey to finish the necessary maps, and Eldridge was called upon to do no inconsiderable part of the topography himself, he being eminently possessed of the topographic skill which forms so important a part of an American geologist's training.

Many other geological investigations in the Rocky mountains were carried on by him during these years, for which he received no public credit, the results having been either not yet published or else incorporated into the work of other geologists.

In 1889, in addition to other work on which he was engaged, Eldridge was called upon by the Director to compile a record of the artesian wells of the United States, and to map the distribution of coal in the arid region. In the then-existing state of knowledge, either of these pieces of work involved an amount of research that few would have been willing to undertake to complete in any limited period; but Eldridge, with his usual energy and cheerfulness, undertook both, and soon delivered to the Director maps showing, respectively, the distribution of coal-developments in the arid region, and the location of artesian wells throughout the United States, with an article on those of the entire world. There seems to be no record of the printing of these reports, but they may be safely assumed to have been the starting-point for the elaborate investigations on these subjects now conducted by special departments of the Survey.

In 1890, while putting some finishing touches on his Denver Basin work, much of which was dependent for its data upon the records of artesian wells, he made his well-known study of the Florence oil-field,[1] which was the first scientific investi-

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[1] *Trans.*, xx., 442–462 (1891).

gation of commercially successful oil-production in the Rocky Mountain region, and a work that has had a most favorable influence upon an industry that has since assumed great importance in the West.

In the decade commencing with 1880, the economic work of the United States Geological Survey was first extended to special investigations of the individual useful minerals, and in the pioneer work of this branch, as of many others, Eldridge took a leading part.

Unusually rich phosphate-rock had recently been discovered in Florida in hitherto unknown geological relations, and a Florida division was created, of which Mr. Eldridge was put in charge. The investigation was particularly difficult, because the country rises but little above tide-level, and no natural sections can be found; hence, the geologist must depend upon artificial excavations for his data. The region, moreover, is unwholesome during the great heat of summer; consequently, the work was mainly carried on during the winter and spring months, and Eldridge continued to give up his summers to work in the Rocky mountains; thus, he accumulated double the amount of field data, and had a correspondingly smaller amount of time than his fellow-geologists had for working up his material in the office. For a long time, moreover, accurate maps of the region were entirely wanting, and, while waiting for these, Eldridge was frequently assigned to work in other fields, which momentarily seemed of more pressing importance; thus, those not cognizant of the facts may have been led to blame him for delays; whereas, in point of fact, he was for a long period able to devote to this work, which he had so much at heart, only odd moments stolen, as it were, during intervals of other work.

In 1893 he published[2] a preliminary sketch of the geology of the Florida phosphates. During the following years, however, he was unable to give any time to this work, being called upon to take charge of a new type of Survey work in the West. This was the rapid geological reconnaissance of hitherto unexplored areas of the Rocky mountains, destined to fill up gaps in the geological map of the United States on the 40-mile

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[2] *Trans.*, xxi., 196–231 (1902–3).

scale, in course of preparation by the Survey. Mr. Eldridge was eminently endowed with the qualifications for such work—namely, energy in exploration and a field-experience that enabled him to grasp rapidly the salient features of the geology of a region. During the summers of 1893 and 1894, he made such surveys of large areas in northwestern Wyoming and in northeastern Idaho, respectively, and in each case had worked up his material, and had his report ready for the printer, before the opening of the succeeding field-season.

In the summer of 1895 he was assigned to the investigation of the mineral resources of the Uncompahgre and Uinta Indian reservations, in Utah, a knowledge of which had become of importance to the government, because of the demand for the throwing open of these lands to public occupation. In that summer's work he investigated, alone and unaided, 7,000 sq. miles of very difficult country, and, as a result of his work, prepared in the following winter a report[3] on the most important mineral resources of that area—the Uintaite-deposits—which constituted at that time the most comprehensive discussion of such deposits, not only for this region, but for the whole continent.

Immediately upon completion of this work, in the spring of 1896, his Florida phosphate-work was taken up again, and carried on continuously until interrupted in the following year by a special order of the Secretary of the Interior, that he make an investigation of the asphalts of Utah, which should furnish data for the Land Office in regard to the best methods of opening up the Indian lands for mineral occupation. His report on this work, being for the private use of the Department, has never been published.

When the discoveries of gold in Alaska had assumed considerable industrial importance, Congress, by its Act of Jan. 28, 1898, ordered special geological and topograpical surveys of that region to be made, appropriating $20,000 for the purpose. This constituted an entirely new branch of work for the Survey, and required a novel and special organization of parties and methods of transportation, which could not be governed to advantage at a distance of 6,000 miles from the field of work, but necessitated a responsible head, who could direct

[3] *17th Annual Report U. S. Geological Survey,* Part I., pp. 909–949 (1896).

the work on the spot. The qualifications for such a position were administrative capacity and tact, combined with geological experience and great physical endurance. Eldridge, being judged to be the one who best combined all these varied qualities, was appointed geologist in charge of this work in February, 1898. He at once took charge of the outfitting and preparation of the various parties, which sailed from Seattle on the United States gunboat "Wheeling" on April 5. He had characteristically chosen for himself what promised to be the most difficult piece of work—namely, the exploration of the highly-mountainous interior of the main peninsula. On May 4, with Mr. Robert Muldrow and several camp-assistants, he started up the Sushitna river, following it to its head, and then crossed the divide on foot to the drainage of the Tanana river. They thus passed near and measured, for the first time, the highest point on the North American continent—Mt. McKinley (20,464 ft.). The amount of hardship endured during the little over four months occupied by this trip, can be appreciated only by those who took part in it, and they were men who are not inclined to complain of such things. For weeks, and even months, they were pushing their heavily-freighted boats against the ice-laden current of this great stream, often above their waists in the icy waters for long periods. Then, when the head of navigation was reached, they were obliged to pack on their backs all their heavy camp-stuff over a route that, because of the bulk of the material transported, had to be gone over at least twice on each day's journey. Eldridge's friends, who were familiar with his rugged vigor, were struck with the change in his appearance upon his return from this venturesome trip. The man, whom physical hardship had hitherto seemed only to render more vigorous and sturdy, at length showed evidence of physical wear. This can hardly be wondered at, for Eldridge was then in his forty-fourth year, and the experience of Survey work in Alaska has shown that exploration in the interior should be intrusted only to young men of rather exceptional vigor, if they are expected to endure more than a single season's campaign. It was at the close of this trip that Eldridge reached the crowning happiness of his life, in marrying Miss Jessie Newlands, of San Francisco, niece of the Senator from Nevada.

His report on the work in Alaska being completed in April, 1899, he at once resumed work on the Florida phosphates, which was again interrrupted by his assignment, at the commencement of the fiscal year 1899–1900, to the study of the asphalts and bituminous-rock deposits of the entire United States. The necessary field-investigation for this work took him into no less than ten States and Territories, and occupied all his time, winter and summer, until the fall of 1900. The results of this work were published[4] in an abundantly illustrated form, and accompanied by maps of all the principal asphalt-fields, practically constituting, by its exhaustive treatment, a standard monograph on this class of deposits.

In the spring of 1901, immediately upon the completion of his asphalt report, Eldridge once more resumed work on the Florida phosphates, and had brought it to within a month or two of completion, when, on the opening of the field-season, he was assigned to another investigation, which was judged to be of greater immediate importance—that of the petroleum-fields of California. He took up this work with his usual energy and thoroughness, temporarily closing his newly-acquired home, near Washington, to live in California while the field-work was in progress. In the fall of 1902 he returned to Washington, to write up his report on this, his last, and, as he thought, his greatest, piece of work. From that time onward, summer as well as winter, with scarcely an interval of rest, he devoted his entire energies to the working-up of the enormous amount of material he had gathered to elucidate the structure of those extremely complicated regions. His many maps and diagrams had been fully completed, the chemical and structural problems discussed, and the most important chapters of his report were not only written, but fully revised, before he was incapacitated for further work by his final illness.

The actual cause of this last sickness will, perhaps, always remain in doubt. That, with his magnificent physique, and a power of endurance that seemed inexhaustible, he should have been taken off in the prime of his manhood, was evidently due to his having overtaxed his powers; and continued confinement to the office, especially during the heats of the Washington

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[4] 22d *Annual Report U. S. Geological Survey*, Part I., pp. 209–452 (1901).

summers, undoubtedly had a debilitating effect upon one so accustomed to an outdoor life as he was.

In October and November, 1904, he was laid up for a month with a peculiar illness that took the form of a low fever and baffled the physicians. In January, 1905, possibly before he was physically capable, he made a trip to Cuba to examine some asphalt-deposits, returning by way of Florida to verify the latest developments in the phosphates. This was his last field-work. Within a month after his return to the office illness again obliged him to give up work. He underwent a painful surgical operation, from the effects of which, at first, he recovered so rapidly that he expected in a few weeks to resume his nearly-completed work. This relief proved, however, to be only temporary. The former physical troubles returned, caused, as it proved, by sarcoma of the kidney, and, after a painful and prolonged struggle, he finally succumbed on June 29, 1905. Through all this last illness the prevailing characteristics of his life, the combination of strength and sweetness of character, showed in strongest relief. Though doubtless conscious before all others that he was near his end, he remained cheerful and courageous to the last, and, when the fatal result could no longer be concealed from his wife, he calmly proceeded to put his worldly affairs in order to the utmost detail.

Although his two last and most important pieces of work, the studies of the phosphates of Florida and of the asphalts of California, were neither of them complete at the time of his death, they will by no means be lost, since much of his manuscript was already written and his material and notes were clearly and systematically arranged, so that they can be prepared for publication with comparatively little supplementary work.

Eldridge was remarkable for his robust physique, his strong and admirable personality, his exceptionally cheerful nature, and the keen interest with which he entered into all pursuits, whether of work or recreation. His early instructor and lifelong friend, Prof. Shaler, says of him:

"He will remain with me as the type of the strong, well-balanced man; brave, steadfast, patient in his duties, ever friendly with his neighbors, helpful with his friends. I feel that my contacts with him served to ennoble my life."

It seems one of the inscrutable provisions of Providence that this man should have suddenly been taken away in the full vigor of manhood, and just as he was nearing the goal to which, in his public as well as in his domestic life, his great energies had so long been directed. But, admired and respected as he was by all who knew him, his energetic devotion to his work and the contagious cheeriness of his personality will ever remain an inspiring memory among his associates.

The following papers have been presented to the Institute by Mr. Eldridge:

Title.	Volume.	Pages.
The Florence Oil-Field, Colorado, . . .	xx.	442–462
A Preliminary Sketch of the Phosphates of Florida, ...	xxi.	196–231

In addition to the above papers, Mr. Eldridge's publications are as follows:

Montana Coal-Fields: *Tenth Census of the United States* (1879–1880), vol. 15, pp. 739–757 (1886).

The Industries of the Base Metals (Lead, Zinc, and Copper) in the Census Year: *Tenth Census of the United States* (1879–1880), vol. 15, pp. 809–830 (1886).

On Some Stratigraphical and Structural Relations of the Country about Denver, Colo.: *Mining Industry* (Denver, Colo.), vol. 3, No. 3, pp. 24–25; No. 4, pp. 33–35; No. 5, pp. 44–45 (1888).

Some Suggestions upon the Method of Grouping the Formations of the Middle Cretaceous and the Employment of an Additional Term in Its Nomenclature: *American Journal of Science*, vol. 38, pp. 313–321 (1889).

On Certain Peculiar Structural Features in the Foot-Hill Region of the Rocky Mountains near Denver, Colo.: *Bulletin of the Philosophical Society of Washington*, vol. 11, pp. 247–274 (1892).

Artesian Wells of Eastern Dakota: *Compte Rendu*, *International Congress of Geologists*, 5th session, p. 318 (1893).

Anthracite-Crested Butte Folio, Colorado (in conjunction with S. F. Emmons and Whitman Cross): *Geologic Atlas of the United States, folio* 9, *U. S. Geological Survey* (1894).

A Geological Reconnoissance in Northwest Wyoming: *Bulletin No.* 119, *U. S. Geological Survey*, pp. 72 (1894).

A Geological Reconnaissance Across Idaho: 16*th Annual Report, U. S. Geological Survey*, pt. 2, pp. 211–276 (1895).

Occurrence of Uintaite in Utah: *Science*, new series, vol. 3, pp. 830–832 (1896).

The Uintaite (Gilsonite) Deposits of Utah: 17*th Annual Report, U. S. Geological Survey*, pt. 1, pp. 909–949 (1896).

Geology of the Denver Basin in Colorado (in conjunction with S. F. Emmons

and Whitman Cross): *Monograph No.* 27, *U. S. Geological Survey,* pp. 556 (1896).

Report of the Sushitna Expedition (in conjunction with Robert Muldrow); The Extreme Southeastern Coast; The Coast from Lynn Canal to Prince William Sound; The Sushitna Drainage Area; Maps and Descriptions of Routes of Exploration in Alaska in 1898, with General Information Concerning the Territory: *Special Publication of the U. S. Geological Survey,* pp. 15–27, 101–102, 103–104, 111–112 (1899).

A Reconnaissance in the Sushitna Basin and Adjacent Territory, Alaska, in 1898: *20th Annual Report, U. S. Geological Survey,* pt. 7, pp. 1–29 (1900).

The Asphalt and Bituminous Rock Deposits of the United States: *22d Annual Report, U. S. Geological Survey,* pt. 1, pp. 209–452 (1901).

Origin and Distribution of Asphalt and Bituminous Rock Deposits in the United States: *Bulletin No.* 213, *U. S. Geological Survey,* pp. 296–305 (1903).

The Petroleum-Fields of California: *Bulletin No* 213, *U. S. Geological Survey,* pp. 306–321 (1903).

Biographical Notice of Edward Cooper.

BY R. W. RAYMOND, NEW YORK, N. Y.

(Bethlehem Meeting, February, 1906.)

EDWARD COOPER was born in New York City, Oct. 26, 1824. His father, Peter Cooper, to say nothing of manifold reasons for fame as an inventor and philanthropist, deserves to be remembered as a pioneer in the establishment of public schools; and it is not surprising that he sent his son to one of the democratic institutions in which he was himself so ardently interested. From the public school Edward Cooper went to Columbia College, where he made the acquaintance of Abram S. Hewitt, who was first his tutor, afterwards his traveling companion, brother-in-law, business partner and life-long friend.[1]

For twenty years I was the consulting engineer of the firm of Cooper & Hewitt, and also connected with the work of the Cooper Union, in which they were both deeply interested and faithfully active. On the basis of the intimate knowledge thus gained, I venture an estimate of Edward Cooper's character and career which, if not fully comprehensive, may facilitate a juster estimate than would be reached by the student of those data which the modesty of the man permitted to be placed upon public record.

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[1] See, for further particulars of their early association, my "Biographical Notice of Abram S. Hewitt," *Trans.*, xxxiv., 186 to 204 (1904).

Edward Cooper's mind was one of the most active, accurate and subtle with which I have ever come into contact. He had inherited from his father a sunny, benevolent temperament, and the irresistible impulse to attack all new problems which challenged his attention. His thorough education had indeed made him wiser than his father, both in philanthropy and in technical inventions. He could condemn and reject a proposition which Peter Cooper would have embraced, in a splendid, though ignorant, faith that nothing was impossible to an American, and that the word "impossible" was simply a label, put upon things reserved for the citizens of the Republic to achieve. The son's greater knowledge led him to see clearly both sides of a proposition, where the father saw but one. Yet it remained his leading characteristic, that he loved to study, criticise and invent, rather than to conclude, adopt and execute. When he had exhausted a question, he was prone to drop it. So long as it called for further inquiries, he was indisposed to settle it by any final decision.

I have often fancied that one who enjoyed, and utilized, the opportunity to pick up and put into practical application the ideas, suggestions and investigations of Edward Cooper, might attain thereby the fame and fortune to which Mr. Cooper himself seemed indifferent.

As an instance of his passion for intense intellectual labor upon a subject which interested him, I remember a copy of the English translation of Prof. Grüner's work on the thermal reactions of the blast-furnace. The book is full of disgraceful errors in the calculation and printing of equations and tables. Having occasion one day to consult Mr. Cooper's copy, I found that he had laboriously corrected it, from beginning to end, simply for his own satisfaction!

Sometimes (as in the cases of the Durham blast-furnace charging-apparatus and hot-blast stoves[2]) he utilized his scientific ability and inventive genius in useful inventions; but I do not think he ever applied for a patent.

He was deeply interested in the attempts of Chenst and others to produce iron-sponge by the direct reduction of iron-oxide ores; and, as early as 1863 or 1864, he instituted at the Andover furnace, Phillipsburg, N. J., experiments in the reduc-

[2] *Trans.*, xiv., 130 to 139, and xxxv., 582 (1905).

tion of iron-ore, without fusion, by means of heated carbon monoxide, forced through a chamber containing carefully selected and broken ores. According to the recollection of Mr. Joseph C. Kent, then and for many years afterwards the manager of the Andover works, this experiment failed by reason of a siliceous dust which, accumulating in the interstices of the finely-broken charge, coated the pieces of ore with a tenacious film, effectually preventing the deoxidizing action of the carbon monoxide. If this was the only difficulty, it could scarcely be regarded as insuperable; and, in fact, Mr. Kent remembers that Mr. Cooper said he could probably devise some way of overcoming it. Probably he dropped the subject for a while. At all events, nothing more was done in that line at Phillipsburg; and in 1867, when the works at Phillipsburg were sold by Cooper & Hewitt to the Andover Iron Co., parts of Mr. Cooper's direct-reduction apparatus were shipped, as possibly available for further experiments, to the Durham Iron Works, Riegelsville, Pa., owned by the firm. Mr. B. F. Fackenthal, Jr., subsequently for many years the manager of the Durham Iron Works, remembers these pieces as embodying novel details (especially in the construction of gas-valves, etc.), which have since been universally adopted. With his usual indifference to personal credit or reward, Mr. Cooper made no claim to them as his inventions.

Meanwhile, he continued to pursue with intense interest and unwearied study the subject of thermal physics and chemistry, particularly as related to the use of gaseous fuel, the Siemens regenerator, etc.; and I remember finding in his office a pile of sheets, covered with figures which represented his elaborate and minute calculations, performed at home and at night (for I never saw him so employed at his office), and antedating much that was subsequently published by others.

As a result of these studies, he began, about 1873, at the works of the New Jersey Steel & Iron Company (Cooper, Hewitt & Co.), at Trenton, N. J., the practical test of a direct-reduction process, which presented many novel features of his own. The principles of this process have been deemed worthy of special recognition by Prof. H. M. Howe, whose classic work contains a description of it, with a diagram illustrating the ap-

paratus.[3] In this process, to quote the preliminary summary there given:

"Iron ore is heated and reduced by a current of hot carbonic oxide, or carbonic oxide and hydrogen. These gases are oxidized to carbonic acid and steam by the oxygen of the ore; they are then passed through a regenerator, in which they are highly heated, and thence through a bed of coal or other fuel, in which they are again deoxidized to carbonic oxide and hydrogen. Still remaining in the same closed circuit, they are then used for reducing a fresh portion of ore, a part of the carbonic oxide and hydrogen, however, being diverted to heat the regenerator already mentioned.

To simplify matters, let us suppose that only carbonic oxide is used, and follow the course of the gas. What is true of pure carbonic oxide would be true of a mixture of this gas with hydrogen, mutatis mutandis."

After giving the calorific calculations involved upon the above assumption, which showed a theoretical excess of heat above the amount required for the chemical reactions, and thus available to make up the loss by radiation, to heat the ore to the temperature of deoxidation, etc., Prof. Howe continues:

"This, in Mr. Cooper's opinion, is not a sufficient surplus. Hence he introduces steam along with the carbonic acid into the regenerator and thence into the gas-producer, thus making water-gas, and thus increasing the quantity of gas available for burning in the regenerator, but without introducing nitrogen into the closed circuit of the reducing system. It may, indeed, be regarded as a mode of making water-gas, which is used while still hot from the gas-producer for deoxidizing iron-ore. The steam is introduced in the form of a jet, and incidentally aids the circulation of the gas through the system."

I forbear to quote further from Prof. Howe's description, which is accessible to all interested students of the subject. Though confessedly incomplete as a statement of the somewhat complicated details of Mr. Cooper's apparatus, it clearly indicates that his scheme comprised many ingenious and novel features, and embodied a wonderfully complete plan for the utilization of the heat of all the chemical reactions of reduction and combustion involved.

Indeed, this theoretical completeness, and the consequent difficulty of constructing and operating the several parts of the experimental plant, proved to be one of the reasons for the final abandonment of the enterprise. Explosions due to leaks in flue-walls, and troubles with valves or other mechanical details, involved discouraging delays and fresh outlays for reconstruction, repair, or new design.

[3] *The Metallurgy of Steel*, 2d ed., vol. i., pp. 275, 276 (1891).

But these experiences, inseparable from such undertakings, did not disprove Mr. Cooper's theory, and would not have been sufficient to prevent him from persevering in his plan. Indirectly, however, they had a greater effect; for they delayed his experiments until the improvements in blast-furnace practice, already accomplished or clearly foreseen by his prophetic eye, forced upon him the conviction that the most perfect "direct process" would be unable to compete with the old method of combined reduction and fusion, producing pig-iron as a material for further transformations. The simple circumstance that a "direct process" for the production of iron-sponge would require rich and pure iron-ore, and could be operated only upon a limited scale as to size of the unit of plant, was fatal to that whole class of processes. Theoretical gains in heat-economy dwindled and disappeared before the enormous savings effected in all departments of the manufacture of pig-iron by increase in the temperature and pressure of blast, in the dimensions and productive capacity of each furnace, and in facilities for the cheap handling of raw material and product.

It was characteristic of Edward Cooper that he did not, like many another inventor embarked upon a darling conception, refuse to recognize the controlling current which was leaving him in an eddy of self-centered revolution. On the contrary, with imperturbable good-nature, he said: "I fancy the blast-furnace is going to be the best thing, after all!" and, abandoning the problem of "direct reduction," upon which he had expended so much time, labor and money, turned his attention to other problems, equally attractive and apparently more practically important. During this last stage of his "direct-process" experiment, I was associated with him as the consulting engineer of his firm; and it was at his request that I communicated to Prof. Howe some of the data for the notice of Mr. Cooper's process, contained in *The Metallurgy of Steel.* I may, therefore, claim the authority of personal knowledge for the foregoing statements.

Mr. Cooper was a public-spirited citizen, and a supporter of all meritorious social and municipal movements. He was among those who supported Mr. Tilden in the overthrow of the "Tweed ring," and in 1879 he was elected Mayor of New York City, eight years before his partner, Mr. Hewitt, received

that honor. Mr. Cooper's administration was hampered by many conditions from which his successors were freed by legislative action. He soon discovered that his power as Mayor was very small; but, instead of making that fact an excuse for sullen inactivity, he accepted the situation with characteristic good-nature, and patiently did what he could.

When his nominees for municipal office were rejected by the political body which then held, under the city charter, the power of confirmation, he smiled, and made other nominations. But he thoroughly exposed, through the reports of his Commissioners of Accounts and otherwise, evils which, under the circumstances of that time, the Mayor lacked the power to attack directly and to remove. In my judgment, his administration planted the seeds of many reforms, of which his successors reaped the harvest. I know that, during that period, as before and after it, not only Mr. Hewitt, but many another conspicuous leader, freely sought and greatly valued his counsels.

It must not be inferred from his habitual gentleness and amiability that he was capable of surrendering his personal convictions of principle and duty. On the contrary, I think I never knew so immovably firm a will as that of this kindly, quiet, modest man, who would neither fight nor yield. I recall an important public occasion, when he stood, against all his political associates, a silent, undemonstrative, unconquerable minority of one; and, if it were permissible, I could give sundry other illustrations, both in politics and in business, of the steadfastness with which, against all persuasions, and at the sacrifice of personal interests, he held to his own convictions of right. Nobody could, and, so far as I am aware, nobody ever thought he could, fasten upon Edward Cooper the least charge or suspicion of unworthy motive. In a time of excited political activity and business competition, in both of which he was a by no means insignificant factor, he kept his honor without stain, either deserved through acts of his own, or undeserved, through the slanders or the doubts of others.

Mr. Cooper became a member of the Institute in 1874; and, although he left his own important contributions to American metallurgical and mechanical practice to be described in our *Transactions* by other writers, he maintained, to the time of his death, a hearty interest in the affairs of the Institute, and a cor-

dial and helpful friendship for its members. During the twenty years of my business connection with him, as the engineer of the firm of Cooper & Hewitt, I enjoyed, through the generous agreement of both partners, a free permission to attend all the meetings of the Institute, and, for about ten years of the twenty, to discharge the duties of its Secretary, provided the interests of the firm of which I had charge were not allowed to suffer. Under this proviso, I consulted one or the other of them, as a matter of form, before going off for a week or more, to attend an Institute meeting; but the matter was invariably referred back to me, for my own decision. I trust that this decision was always rendered with due regard to the interests of these generous employers. At all events, the fact that, out of ninety meetings of the Institute, I have been present at eighty-six, and that I attended, without objection, before or after the event, from either Mr. Cooper or Mr. Hewitt, all but one of the meetings which occurred during the twenty years of my business connection with them, is certainly significant.

Moreover, the firm leased to me, as Secretary, at the nominal rent of $800, practically two floors of one of the buildings occupied by their business. In the interior reconstructions involved in this arrangement, Mr. Cooper took the liveliest interest. The arrangement itself was continued for some years after I left the service of the firm, and was terminated only in 1899, when the work of the Institute demanded the removal of the Secretary's office to the commodious quarters which it now occupies.

I venture to relate here an incident, trifling in itself, but thoroughly characteristic of Edward Cooper. One day, after going over with me some matters of business, he said suddenly: "Raymond, I notice in one of your circulars a statement of the cost of life-membership. Now, I have been figuring on that matter; and I find that, in view of my age and normal expectation of life, the present value of an annuity of $10 would be less than the life-membership fee. Consequently, I do not consider life-membership as a good business investment. However, if the Institute wants the money, that is another question!" I assured him that the Institute did not want the money; and he lived thereafter long enough to pay in annual dues much more than the life-membership fee, with compound interest!

But perhaps the most characteristic service rendered to the Institute by Mr. Cooper is recognized in the following minute, adopted by the Council, March 13, 1905:

"Hon. Edward Cooper, who died Feb. 25, 1905, in the 81st year of his age, had been for over thirty years an honored member of this Institute, having joined it, together with his partner and brother-in-law, Hon. Abram S. Hewitt, at a time when the encouragement and support of such men were vitally important to it.

"The pages of its *Transactions* show the value of his contributions to progress in the metallurgy of iron, and his personal interest in the success of the Institute was evinced by his discharge, for many years, of the important, though not publicly proclaimed, duties of a member of its Finance Committee. To many of its members he was a cordial and helpful friend.

"His eminent services in national and municipal affairs, and his upright, generous and winning personal character, enhance both the just pride and the inevitable sorrow with which all who knew him in any of the manifold spheres of his beneficent activity now receive the tidings of his decease."

I treasure a pathetic letter from Mr. Cooper, written not long before his death, in which he requested to be released from the unostentatious but onerous duty of examining and certifying my monthly vouchers, on the sole ground that failing eye-sight prevented him from discharging it.

In common with a host of those who enjoyed the privilege of an acquaintance with Edward Cooper, and as one of the smaller number who were blessed with his daily companionship, I thank God for a life so pure, so unselfish, and so greatly useful to mankind.

Biographical Notice of Alexander B. Coxe.

BY R. W. RAYMOND, NEW YORK, N. Y.

(Bethlehem Meeting, February, 1906.)

ALEXANDER BRINTON COXE was born in Philadelphia, Pa., Jan. 19, 1838, the second of five sons of Hon. Charles Sidney Coxe and Ann Maria Brinton. A more extended history of his family and its important relations to the development of the United States will be found in my Biographical Notice of his younger brother, Eckley B. Coxe, published in 1895.[1] It is therefore sufficient to say here that Dr. Daniel Coxe, a dis-

[1] *Trans.*, xxv., 446 to 476 (1895).

tinguished physician and surgeon of London, and a medical attendant of Charles II. and Queen Anne, purchased, in 1684 and 1686, lands in East and West Jersey from grantees of the Duke of York, and became Governor of West Jersey, in which capacity he did much to develop the provincial fisheries, the manufacture of marine salt and pottery, the exportation of timber and the West India trade. In 1698, having acquired the royal patent of the Province of "Carolana," covering (with some reservations) the territory extending from the Atlantic to the Pacific, between the 31st and 36th parallels of N. latitude, Gov. Coxe sent an expedition from Charleston, S. C., to the Mississippi. In 1769, seventy years later, his grandchildren surrendered to the Crown their title under this vast and vague patent, receiving in return a grant of 100,000 acres in the Colony of New York.

The line of his descendants, so far as they concern this sketch, is as follows:

1. Col. Daniel Coxe, his son, b. 1673, who came to New Jersey in 1700; resided there until his death (1730); held many high offices; and issued in London (1722) *A Description of the Provinces of Carolana*, containing probably the earliest published plan of political union for the British Colonies in North America.

2. His son, William Coxe, b. 1723, d. 1801, an enterprising colonial merchant and trader.

3. His son, Tench Coxe, a leading political economist and statesman of the period immediately following the American Revolution; delegate to the Continental Congress; Hamilton's Assistant Secretary of the Treasury; the introducer of the Arkwright loom; the earliest and most influential advocate of the production of cotton in the United States; the prophet of the use and value of coal as fuel; and the purchaser, in reliance upon his own prophecy, of large tracts of coal-land in Pennsylvania.

4. His son, Charles Sidney Coxe, b. 1791, d. 1879, a distinguished lawyer, at different times District Attorney, and District Judge in Philadelphia, who devoted himself, outside of his professional duties, to the preservation and administration of the coal-lands of his father's estate, and who educated his sons with special reference to the future conduct of the great development foreseen by him.

Alexander, the second of these sons, is the subject of the present sketch.[2]

Alexander Brinton Coxe was educated first at the classical school of Dr. Faries, recognized for more than fifty years as the best of its class in Philadelphia. A good student, like all his brothers, he was able at the age of fourteen (1852) to enter the University of Pennsylvania, where he was graduated in 1856. Fond of open-air exercises, and especially of rowing, he pulled an excellent oar in the University Barge Club. After graduation, and with a view to the future duties of his life, he spent two or three years in a Philadelphia counting-house of the old school, where the traditional methods of commerce and trade were rigidly observed and taught. Of course, these old fashions were already giving way to newer ones, under the pressure of altered conditions; and Mr. Coxe, conducting in after years a business of production, transportation and trade, the extent, complexity and intense activity of which would have paralyzed the energies and systems of his first employers, may have smiled as he recalled their maxims and methods. Yet, in business as in politics, a thorough knowledge of the old is the only safe basis for an intelligent judgment and choice of the new. At all events, the subsequent honorable career of the firm of Coxe Brothers & Co. exhibited, through all modern novelties of organization and operation, the old-fashioned virtues which no amount of progress can afford to discard.

As a further preparation for his life-work, Mr. Coxe made, at the age of about twenty-two, an extended tour in Europe, returning from which, soon after the beginning of the War of the Rebellion, he entered the Union army as an Aide upon the staff of Major-General Meade, who highly esteemed his character and service.

In 1865, the firm of Coxe Brothers & Co. was formed for the development of the anthracite-lands inherited from their grandfather, Tench Coxe, and preserved, with prescient labor and sacrifice, by their father. For forty years (until, a few weeks before his death, the property of the firm was transferred to the

[2] For many of the particulars of this sketch, I am indebted to Mr. John Cadwalader, of Philadelphia, a brother-in-law of Eckley B. Coxe. Indeed, I might have copied *verbatim* the excellent account sent me by Mr. Cadwalader, but for my desire to interpolate personal knowledge and comments of my own.

Lehigh Valley Railroad interest), Alexander Coxe devoted himself unremittingly to that great business. Three of his brothers and a cousin (Franklin Coxe), who had constituted the original firm, successively died, leaving him, at last, to carry alone the burden of this immense responsibility. His brother Eckley, who died in 1895, had always been, not only the technical manager of the colleries operated by the firm, but also the most evident and eminent representative of its position and policy. Fortunately, he had lived long enough to settle many questions involved in his own peculiar department of administration, so that, for the succeeding ten years, the skillful engineers of the house were doubtless able to take his place. But during his life-time, no less than after his death, the all-important financial management and undertakings of Coxe Brothers & Co. were chiefly directed by his brother Alexander. And I had occasion to know personally that the two brothers consulted freely, and acted in perfect harmony, upon all subjects connected with any part of their business.

One of the most important of these subjects was the relation of the firm to its working employees. In my Biographical Notice of Eckley B. Coxe, already cited, I have described and discussed at some length the principles and the policy of the house in this respect; and that statement need not be repeated here. But I then realized that, even in eulogy of the departed, I ought not to ignore the merits of the living; and I am now glad to find that I expressed that feeling in the following foot-note,[3] which I regard as worthy of repetition:

"It is impossible to separate, in such matters, the part taken by the family as individuals from that of the company which was their business representative, or the part of Eckley B. Coxe himself from that of the kindred who so heartily united with him in every good work. While I comply with their own desire, as well as with the general rule of justice, in ascribing to him the credit for the undertakings of all kinds in which he was, so to speak, the official leader, I cannot forbear to say here, once for all, that I do not believe he could have accomplished, and I scarcely believe he would have undertaken, so much, without the cordial and effective support of his wife and his brothers and sisters. This qualification does not in the least detract from his fame; and, on the other hand, it furnishes the assurance that his wisely benevolent schemes and policies will not end with his death."

The expectation thus expressed has been fully realized in the years which have since elapsed.

[3] *Trans.*, xxv., 467 (1895).

Soon after entering upon active business, Mr. Coxe married Sophy, daughter of Richard Norris, of Philadelphia, who, with a married daughter (Mrs. Charlton Yarnall) and four grandchildren, survives him. But no account of his life would be complete without mention of his son, Daniel, who bore the name, and inherited the ability, of distinguished ancestors, and in whom a father's affection and ambition were centered. Unfortunately, this only and gifted son was handicapped in youth by physical frailness, which required unremitting care, change of climate, etc. As he grew older, he became stronger, and was able to exercise effectively his exceptional taste and talent for mechanical engineering. He designed and constructed a locomotive which was highly approved by expert railroad-engineers; and he constructed a narrow-gauge track, upon which practical tests of his invention could be made. His models were exhibited at the Chicago Columbian Exhibition in 1893. It was a doubly cruel blow to his proud and loving father, when this promising son, after surmounting the perils and drawbacks of physical weakness, was killed by the accidental upsetting of his own engine upon his own track. Concerning such a bereavement, I can say nothing, because I know so much. But I may be permitted to bear witness to the encouragement and help derived from the example of a father, thus stricken and stripped, who still recognized the claims of duty, and, with courage and patience, "endured to the end."

Alexander B. Coxe was by no means limited in his sympathies and activities to the sphere of his own business. He occupied many positions of trust, among which may be named those of director of the Lehigh Valley R. R. Co., the Pennsylvania Co. for Insurance, etc., and the old Mutual Assurance Co. (familiarly known as "The Green Tree"), the duties of which he faithfully discharged. In social matters also he was an influential participant, securing by kindliness and courtesy the good-will of all.

Until within a very few days of his death, he appeared to be in excellent health, though he had undoubtedly overstrained his energies in recent labors, and had thus lost the power of resistance to a sudden attack of disease. So it came to pass that he succumbed to a severe cold, developing into pneumonia —the malady so fatal to patients advanced in years, and the

one which, according to high authority, still remains beyond the comprehension or control of modern medical science—and, after a brief illness, died Jan. 22, 1906, leaving behind him a multitude of mourning friends, and the memory of a long, blameless, fruitful life.

Mr. Coxe joined the Institute in 1880, and, although fully qualified to be a Member, modestly preferred to receive, and to retain for twenty-five years, the title of Associate.

A Simple Rotary Distributor for Blast-Furnace Charges.

BY DAVID BAKER, PHILADELPHIA, PA.

(London Meeting, July, 1906.)

In a paper presented to the American Institute of Mining Engineers, September, 1904, entitled "Improvements in the Mechanical Charging of the Modern Blast-Furnace,"[1] I showed the great fault of mechanical charging-devices to be that the materials charged by the dumping of the skip were separated, the coarse materials having high velocity, and the fine, low—thus giving paths of low resistance through the stock in the furnace, which resulted in unequal distribution of the furnace-gases, unequal reduction, slips, scaffolds, irregular cutting of the inwall, "off" iron, and increased fuel-consumption. I showed how this could be prevented by rotary distribution, through which the irregular distribution in one layer charged is compensated by charging the subsequent layers from successively varied positions of the charger, and becomes practically eliminated when the angle between two successive positions is a little more or less than an even fraction of a full circle. Of all the devices then on the market, the nearest approach to the ideal arrangement was found in the Brown distributor, a full description of which was given. The one defect pointed out, however, was the difficulty of maintaining so much mechanism on the top of a blast-furnace.

Since the presentation of that paper, I have designed a very

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[1] *Trans.*, xxxv., 553 to 575 (1905).

simple form of rotary distributor, which may be applied to any form of double-bell charger, retaining the perfect gas-seal, which is so admirably obtained by that construction.

In seeking protection for this invention, I found no similar distributor on record in the patent offices of the various countries where application was made; but an application showing the same construction was filed in the United States shortly afterwards by Mr. Albrecht B. Neumann, of Chicago, Ill. A joint ownership of the invention was the result; and the apparatus has received the name of the Baker and Neumann distributor.

Fig. 1 shows the top of a modern skip-filled furnace, provided with a main bell, closing the mouth of the furnace, and a gas-seal bell above, closing the gas-seal. It will be noticed that this small bell, which in ordinary construction permits a discharge of the materials all around its periphery, is here provided with a deflector-plate that causes the material to discharge around only one-half of its circumference, when the bell is lowered. The dotted lines show the position of the distributing-bell and plate when the load is discharged.

Rotation is given to the bell and plate during its upward movement to close, by the ratchet-lever (*A*), which drives the bevel-gear (*B*), through which the hollow rod (*C*) passes, but to which it is rotatably connected by two wings or feathers, projecting from opposite sides of the hollow bell-hanger and traveling in corresponding grooves in the hub of the bevel-gear (*B*).

Fig. 2 is an enlarged view of the upper end of the hollow distributing-bell hanger, the crosshead connection to the bell-operating beam and the bevel-gear driver, showing also the ratchet-lever (*A*) and its method of attachment to the bell-operating lever, with the connection of the indicator-rod leading to the ground.

By drilling holes where required, the ratchet-lever may be provided with several points of attachment to the bell-beam, and the revolution of the distributor set at any angle desired. The ratchet permits the lowering of the distributing-bell without imparting any turning movement to the plate and bell; but the rotation is made during the closing of this bell, when it is without any load.

This construction can, therefore, be made strong and simple,

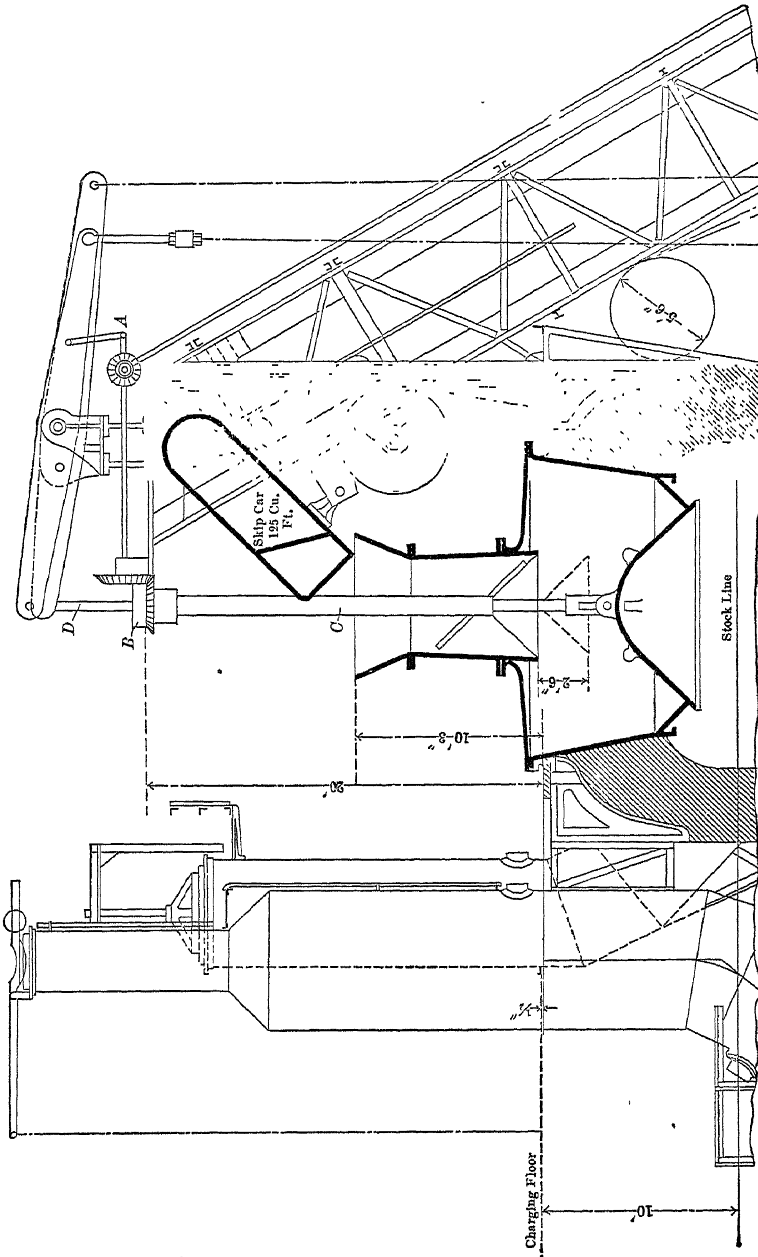

FIG. 1.—TOP OF A MODERN SKIP-FILLED FURNACE.

with few parts, requiring very little attention, and practically as simple and durable as the bell-beams. The preferred method of operation is to send the fuel of the charge to the top in four

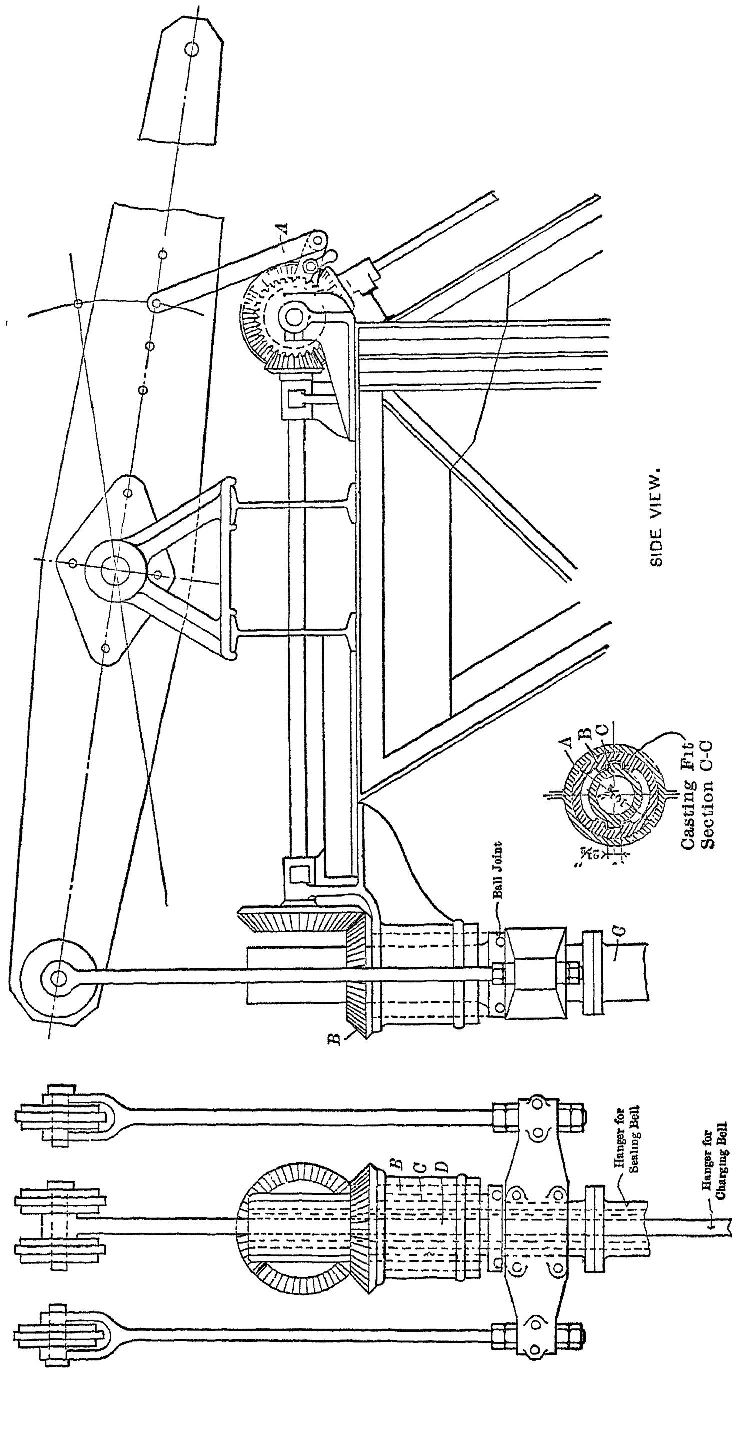

Fig. 2.—Enlarged View of Ratchet-Lever.

skips and the ore and limestone in the following four skips; but in some localities it has been found advantageous to hoist the flux and coke together.

When the whole charge is thus hoisted in eight skips, the distributing-bell is set to rotate 91° at each discharge, which causes the center points of each discharge of the distributing-bell to rotate 8° for each charge from the position of the preceding charge. The overlapping corrects the sorting which may have been made in any one charge.

It is possible, when desired, to arrange the throw of the distributor a little more than 120° or 180°, depending on how the furnace-charge is to be hoisted, but I prefer to hoist the charge in eight skips and give a 91° movement to the distributor.

At the present writing, seven furnaces have been equipped with this device, the first one having been installed a little over a year ago. As was expected, the mechanical part of the arrangement has given no trouble whatever. A door is provided, however, in the receiving-hopper, through which, by means of a steel bar and sledge, the plate may be cut loose in a few minutes, and dropped into the furnace. Fortunately, this safeguard has never been needed.

The results of the operation of this method of distribution have been very gratifying. The work of the three furnaces in which the device was first installed, compared with their previous record, shows an average saving in fuel of 5.4 per cent., an increase in production of 21.6 per cent., a large reduction in the flue-dust thrown off, and a great increase in regularity of running.

The Gas-Producer as an Auxiliary in Iron Blast-Furnace Practice.

BY R. H. LEE, LIBERTY FURNACE, VIRGINIA.

(London Meeting, July, 1906.)

WITHOUT doubt, one of the most frequent and serious annoyances connected with the practical running of a blast-furnace, especially in single-furnace plants, is caused by low steam, in spite of the fact that all boilers at blast-furnaces have grates for burning coal, and that coal is more or less continuously burned upon them, from three to a dozen men being pretty constantly engaged around the boilers as firemen or coal- and ash-handlers.

At times, when the furnace is "tight," all available men are put on to assist the regular firemen; yet sufficient steam cannot be got to keep the engines up to their regular number of revolutions, while the stoves, deprived of the gas which has been sent to the boilers, run cold, so that, when the furnace loosens up and resumes the regular rate of driving, the blast is very considerably below normal temperature, and a "cold," or at best a "high-sulphur," cast is the result. Again, it often happens that a furnace, running at its best, becomes temporarily a little too hot: the pressure rises; the amount of gas made is diminished; the steam soon commences to go down; and a small scaffold is likely to form before the furnace has been brought back to the proper rate of driving, and thus enabled to take its regular amount of blast. The usual remedy is either to "jerk" the furnace by a sudden, brief intermission in blowing, or to cool the furnace by lowering the heat of the blast; but if the lower blast-temperature be maintained too long, the hearth and descending stock may be cooled below the "critical temperature" mentioned by Mr. Johnson,[1] and the furnace may go to the other extreme, and become cold. This is especially the case when coal or coke high in sulphur is used.

[1] Notes on the Physical Action of the Blast-Furnace, *Trans.*, xxxvi., 454 to 488 (1906).

Possibly a day or more may be consumed in getting the furnace back to normal conditions.

The great value of an abundant blast-capacity in the handling of a furnace was once brought home to me in a striking manner, in the management of a furnace of 250 tons capacity, working on soft coke and a very fine brown hematite, similar in structure and fineness to the Lake Superior Mesabi ores. It was one of a plant of three furnaces; but the gas-mains of the furnaces were not connected. Consequently, when steam was low on one furnace, no help could be obtained from its neighbors, and the engines of this particular furnace were always run at their full capacity. The soft coke and fine ore gave rise to more or less sticking, which was handled in the usual way. During the campaign, the blast-main from the spare engine of one of the other furnaces (95 by 21 ft. in size) was connected with this furnace, so that, while the latter depended ordinarily upon its own boilers and engines, the spare engine of the large furnace could be at once put on, if the pressure of blast began to rise. Since, however, the loss of blast from unavoidable leakage around a furnace increases rapidly with the pressure, it was not sufficient to add merely enough blast from the spare engine to make up for the loss measured by the revolutions of the regular engines; much more than this theoretically necessary amount was always needed. The normal amount of blast used was about 28,000 cu. ft. per min. at from 11 to 12 lb. pressure. The spare engine, a double compound Tod of 30,000 cu. ft. capacity, was generally run at about 20 rev. per min., thus giving about 15,000 cu. ft. extra. As a rule, if the foreman put on the extra engine as soon as the gauge showed the pressure of the blast to be increasing, 20 min. sufficed to loosen the stock and get the furnace back to its normal "gait;" but if, as was sometimes the case, the pressure was allowed to go up 3 or 4 lb. before it was possible to furnish this relief (the spare engine being at work on its own furnace), then one and sometimes two hours of hard blowing were required. The usual remedies of "jerking" and cutting-down the heat of the blast were discontinued; and the only one used afterwards was the very simple procedure of turning on the spare engine for a short time, upon which the pressure gradually fell to the normal, when the extra engine was taken off.

Under this treatment, the furnace was not only more quickly loosened up, but, since the rate of driving had not been much reduced, the tonnage suffered but little.

It is true that, under the various conditions of practice, all furnaces do not behave in the same manner, so that adding more blast for a short time may not prove successful in all cases; but situations in which an abundant reserve of blast would be of advantage occur very often, as every furnace-man knows.

A large number of boilers of the best modern water-tube types will not effect this result, because, as soon as the gas-supply to a boiler is reduced, the steam must fall from lack of fuel. Firing with coal, of course, helps to a certain extent; but it does not take the place of abundant gas. Forced draft, with hand-firing, although better than hand-firing with natural draft, is not as efficient as plenty of gas. The automatic stoker avoids the cooling due to frequent opening of charging-doors involved in feeding coal by hand; but it is impossible to push the stokers beyond a certain speed, so that, except in the amount of labor required, the machine has no particular advantage over hand-firing with forced draft, for the rapid raising of steam in an emergency. Moreover, it is useless to add boilers to a furnace-plant beyond a certain limit, because, unless there is sufficient gas to fill the combustion-chambers properly, no advantage is gained. On the contrary, the results are not as good as those attainable with fewer boilers and more gas to each one.

It seems necessary, therefore, to adopt some other means for supplying additional blast at need; and I can see no cheaper or more feasible means than the use of a gas-producer driven by forced draft.

With producers, the amount of coal burned in good practice would not vary much from what is now used. Even the light fires kept on the boiler-grates to keep the gas lit might be dispensed with, since the flow of gas to the boilers would not be checked, and there would be no danger of the flames going out. The net calorific effect of coal burnt under the boilers and in the gas-producers, respectively, is, if not the same, rather in favor of the producer; indeed, the efficiency of carbon burnt in the form of producer-gas is claimed to be from 5 to 25 per cent. greater than that of direct combustion of solid fuel. It is certain, therefore, that no more coal would be used than in

the present way, while the producer would give the added advantage of permitting at all times a perfect control of the amount of gas under the boilers. Moreover, a glance at the following average analyses of producer- and furnace-gases shows the greater richness of the former, and the great advantage of having them available, as a portion of the gaseous mixture burned under the boilers.

Average Analyses of Producer- and Furnace-Gases.

	A. Producer-Gas. Volume Percentage.	B. Furnace-Gas. Volume Percentage.
Carbon monoxide (CO), ...	22.0 to 30.0	25.0
Carbon dioxide (CO_2), ...	6.0 to 1.5	13.5
Hydrogen (H), ...	15.0 to 7.0	1.5
Methane (CH_4), ...	3.0 to 1.5	0.0
Nitrogen (N), ...	54.0 to 60.0	60.0
	100.0 100.0	100.0

A, from R. D. Wood & Co.'s pamphlet on Producer-Gases; B, from 50 analyses made under the writer's direction.

The arrangements for delivering the producer-gas to the boilers, and the size and form of burners used, would naturally vary according as local conditions demand. Either overhead or underground mains could be used, provided there were proper cleaning-facilities. Separate mains for the blast-furnace gas and the producer-gas should be employed: (1) in order that the producer-gas main could be cleaned without shutting down the furnace; (2) to prevent the producer-gas from flowing up the down-comer to the top of the furnace, instead of being drawn under the boilers. This would occur only when shutting off the blast, and would be due to insufficient height of the boiler-chimney, though otherwise this might be as high as present practice demands. The latter risk could be obviated by means of suitable valves placed between the boilers and the furnace; but such valves would have to be closed at every step, which would be troublesome, and therefore might sometimes be forgotten.

The greater cost of separate mains and separate burners at the boilers might be advanced as an argument for turning the producer-gas into the blast-main from the furnace to the boiler-plant; but I believe the two objections above noted would, in

the long run, more than counterbalance the saving in first-cost resulting from using a single system of mains for both gases.

The use of producer-gas in the stoves, as well as under the boilers, is not suggested as one of the advantages to be secured by having a few producers connected with a blast-furnace. If desired, the producer-gas could as easily be introduced into the stoves as under the boilers; but its use there would be so infrequent as hardly to warrant the expense of installation. The real advantage of adding producers to a blast-furnace plant would be confined to the boilers, where the gain from having an abundant supply of rich gas, under perfect control, would very soon repay the cost of installation by increased output and greater regularity of product.

To a plant of three or more blast-furnaces, gas-producers would not be of as great value as to a single furnace; yet even with three or four connected furnaces, coal must be burned under the boilers; and in this case, also, the gas-producers would have the advantage over the present mode of coal-firing, that a greater calorific effect is obtained from the fuel. In either case, the labor would be a trifle less, and the firing would be under better control, which would mean less variation in the amount of air blown through the furnace, with all the well-known attendant advantages of such regularity.

Internal Stresses and Strains in Iron and Steel.

BY HENRY D. HIBBARD, PLAINFIELD, N. J.

(London Meeting, July, 1906.)

I. INTRODUCTION

A NOTED ordnance engineer once said to a friend, in speaking of the production of great steel guns, "How is it? We design our guns with a factor of safety of eight, and the guns burst."

The vague way in which internal stresses and strains in iron and steel are often considered and spoken of makes it worth while to examine them, as to their nature, their causes and results, how they may, for useful purposes, be advantageously dealt with, and how, when detrimental or dangerous, they may be reduced or kept within harmless proportions.

Rankine defines "strain" as a change of form or dimensions of a solid or liquid mass produced by a stress. This definition seems intended not to cover strains due to internal stresses, and it will help us in the present inquiry to consider strain as a tendency to change as well as an actual change of form.

Internal strains in iron and steel are the result of stresses within the mass of the piece, some parts pulling and others pushing in resistance to them; or, in other words, some parts are in tension and some in compression, each part striving to relieve itself from strain and make the piece assume a form in which all parts are at rest. Subdivision, so that each part could relieve its strain by motion relative to the other parts, would, if carried out far enough, practically obliterate strain.

The internal strains we are now to consider are not those due to the stresses of the service which the piece is rendering, either alone or as a part of a structure, which may be termed service-strains, but those which originate or at least are contained in the piece itself.

Internal stresses in a piece of metal are, like action and reaction, always equal and in opposite directions. The stresses

of every part in tension are balanced by those of other parts in compression. If a part chiefly under tension is removed, as by cutting it away from the piece, the remainder will at once adjust itself so that the stresses are equalized, and a part which has been under compression will become under tension, while the total amount of stress in the piece will be reduced, accompanied by a change of shape, because the remainder of the piece will not have then the same amount of compression- and tension-stresses. Cutting away metal along the neutral axis of a bar, however, as, for instance, by drilling a hole through the center of a flat cold-rolled bar, often may be done without causing an appreciable change of shape.

The importance of internal strains depends on their intensity as well as on the ductility of the metal. Strains which could be allowed with impunity in a ductile or tough steel might be fatal to the integrity of the piece were the metal a hard, brittle variety.

Strains exist in all pieces of cold iron and steel, but as the term is generally used it refers only to such as are made apparent in some way, such as causing a change in the physical properties, rupture, danger of rupture or change of shape of the piece.

Internal strains reduce the specific gravity of a piece of iron or steel. Consequently, the metal is heaviest when it is in the annealed state, meaning that variety of annealing which is the result of heating to redness followed by slow cooling. It is a common error to assume that a piece of steel which has been cold-hammered or cold-rolled or wire-drawn is "more dense" after these operations than it was before, when it is actually lighter.

Internal strains accompany if they do not indeed cause inferior resistance of the metal to chemical action. A file broken off, and the broken end immersed in dilute hydrochloric acid, has been found to have lost a greater thickness of metal from the hardest part, at and near the outside, in which the strains are greatest, than from the less hard central portion. The quick corrosion of wire-nails and fence-wire may possibly be ascribed to the cold-worked condition, and directly or indirectly to the internal strains. The corroding-agent more readily enters the mass of the metal, it may be, because of larger intermolecular spaces.

For the purpose of this paper, internal strains in iron and steel may be divided into two classes, according to the causes which produce them, viz.:

1. Those caused by an irregular rate of change in temperature—that is, of heating or cooling.
2. Those caused by cold-working or permanent change of shape of the piece under consideration by mechanical means at atmospheric temperatures, or at least at temperatures below that at which the metal is softened.

II. Internal Strains Caused by an Irregular Rate of Change of Temperature.

This class of internal strains is by far the more important of the two.

Because iron expands when heated and contracts when cooled, and indirectly for other reasons, hereinafter noted, internal stresses, and therefore strains, varying in all degrees from harmless to fatal, exist in all pieces of iron and steel. This arises from the fact that all commercial iron and steel is the product of processes involving heat, the strains in most cases being those set up by the more or less irregular rate of cooling.

The rate of change of temperature may act in a purely physical manner in producing strains, as will be considered later; or, when it comprises quenching a piece of steel from a high temperature, it may have also a chemical effect on the metal which, by changing the properties thereof, reducing ductility and increasing elastic limit until that perhaps coincides with the ultimate strength, exercises a potent effect on the resulting strains.

The amount of change in temperature does not affect the strains produced. The rate determines all.

The strains due to irregular cooling, unless chemical changes are involved, are apparently between part and part. Those due to cold-working may be considered as between molecule and molecule because of the disarrangement of the molecular formation. The latter results usually in strains between the parts as well.

Strains which come under this division of the subject should be considered under two aspects: (1) those which arise during the continuance of the causes which produce them—that

is, temporary strains—and (2) those which remain after their causes have ceased to act—that is, permanent strains. Temporary strains in many cases become permanent, especially when set up during cooling at an irregular rate. The phenomena of permanent strains are, during their formation, the same as those of temporary strains.

The following cases will illustrate the two ways in which each kind of strain may result in rupture of the piece. When an ingot of hard steel is placed in a red-hot furnace, and is so ruptured internally by the faster expansion of the exterior, due to the rapid heating it undergoes, that it separates into pieces when forged or rolled, its ruin was caused by strains occurring while the cause—namely, the quick heating—was in operation. This is a typical example of temporary strains in steel. When a boiler-plate of soft steel, lying cold on the floor of the shop, suddenly cracks, it is because of strains existing after their cause had ceased its action. This is the usual kind of internal strain occurring in iron and steel, and is the kind chiefly meant in what follows in this part; it is an example of permanent strain. The two instances cited are, it will be understood, extreme cases, in which the piece is ruptured by the intensity of the strains.

Strictly speaking, permanent strains are but relatively permanent, since they decrease when the piece is again heated, or perhaps through the seasoning or annealing action of time. When strains result in rupture of the piece of metal, they are thereby much reduced in amount and otherwise modified.

While strains are unavoidable, they are—except in special cases—undesirable in proportion as they are great. The exceptions arise when the strains increase desirable properties, as, for instance, hardness in hardened steel.

Expansion or contraction of iron or steel by change of temperature does not, if the new temperature has become uniform throughout the piece or structure, set up new permanent strains or materially change those existing between the different parts of the piece or structure, provided that the different parts are all of the same kind of metal, or at least of metals having the same coefficients of expansion. In a structure made of metals having different coefficients of expansion rigidly fast-

ened together, a change in temperature is sure to change the strains, increasing or decreasing them.

The intensity of the strains considered in this part depends on the following determining factors:

1. The rate of change of temperature;
2. The shape of the piece;
3. The bulk or volume of the piece;
4. The elastic limit of the metal;
5. The ductility of the metal;
6. The coefficient of expansion of the metal.

1. The rate of change of temperature is the great governing condition producing or reducing internal strains, both temporary and permanent. The faster the rate, the greater the strains. The amount of change, however great, is unimportant in this connection if the rate be sufficiently slow.

The rate depends on the difference between the temperature of the piece of metal and that of its environment, and on the heat-conductivity of the metal. The greater the difference in the temperatures mentioned, the greater the strains resulting, and the greater the heat-conductivity, the less the strains. A poor heat-conductor, like manganese steel, must, if of massive form, be heated very slowly if dangerous strains are to be avoided.

Extreme cases in which the rates of change of temperature are the greatest met with in the practical manipulation of iron and steel are, for rising temperature, when the article is placed cold within the hot heating-chamber of a furnace, and for lowering temperature, when it is taken heated from a furnace and plunged into a cooling-bath.

With the exceptions noted later, any change in temperature up or down below the degree of heat at which it softens, produces in a piece of iron a somewhat irregular rate of heating or cooling, and hence alters the strains; because, as heat must enter and leave the article through its exterior surface, the portions of the article adjacent to the heated or cooled surface will be warmer or cooler than the parts farther away from such surface. Time is required for heat to pass from one part of the article to another, due to imperfect conductivity of the metal. It follows, therefore, that during a change of temperature

strains are set up or diminished within the piece, through the differences in expansion and contraction, due to differences of temperature of the parts. These differences vary in degree with the rate of change in temperature, being the greater for a given iron or steel article the more rapid such rate of change.

To avoid danger to the piece from this cause, which is the one to which may directly be laid nearly all the actual damage done by strains in iron and steel, one must maintain less difference between the temperature of the article and that of its environment. If the dangerous strains arise in heating the article, it must be heated more slowly. The heating-chamber or receptacle must not be too much hotter than the article placed therein. In extreme cases, the chamber and the article must be slowly heated together, this procedure being called for, however, only in the heating of bulky articles of very hard iron or steel.

When dangerous strains are liable to arise from too rapid cooling, they may sometimes—as in the case of a cast-iron carwheel—be relieved or reduced in intensity by a slower rate of cooling, the red-hot wheels being piled up in tight soaking-pits built in the ground, and allowed to cool very slowly, consuming several days in the operation. The lowering of the temperature of both the thick and the thin parts is maintained at a rate sufficiently uniform to insure their reaching ordinary atmospheric temperatures at about the same time; the thicker parts do not then continue their contraction after the thinner ones have ceased theirs, which action, if it took place, would result in strains that might easily be dangerous to the integrity of the wheel, in view of the almost total lack of ductility in the cast-iron of which it is made.

Many steel castings may not be allowed to cool in the open, or even in the sand, without danger of spontaneous rupture from internal strains, but must be placed while still hot in a heated receptacle, usually a furnace, and allowed to cool with the furnace at a much slower rate than if in the open air. Steel wheels, as well as the iron carwheels mentioned above, usually require such treatment, though the ductility of the softer grades may admit of its omission. On the other hand, the greater coefficient of expansion of steel tends to set up greater strains than those of the iron wheels.

When, however, the rate of cooling may not be reduced—as in quenching, in the heat-treatment of steel, where it must be rapid to give the metal the properties desired—strains may be kept within the danger-point only by having recourse to the modifying conditions which are noted later, and chiefly those relating to the shape and bulk of the piece and methods of heating and cooling.

In special cases—for example, in cutting-tools of steel where only the cutting-edges demand the rapid change of temperature—some relief, often adequate, is to be afforded by heating only the cutting-edges themselves to the hardening-heat. Then the remainder of the article will not have to undergo such a rapid change of temperature as if made as hot as the hottest part, and the resulting strains in the tool will be much lessened and perhaps made harmless.

If a piece of cold iron or steel of uniform temperature, in which the strains are the result of cooling, be cooled further, uniformly as to the exterior surface, the strains will be temporarily reduced and may be wholly eliminated during the cooling operation, only to be restored much as before when the temperature of the piece has again become uniform. Such a piece of steel presents during the further cooling the only exception to the statement that all pieces of cold iron and steel have internal strains. A further practical exception is found in the case of a piece of cold iron or steel which has been cooled from a red heat, at which it is too soft to have strains, at an exceedingly slow rate, as in the case of the carwheel mentioned. When the slow rate of cooling is actually continued to atmospheric temperature the article is practically free from strain.

When a piece of iron or steel containing intense strains is heated in one part gently—it may be to 200° or 300° F.—the strains are changed so that when the piece is cold it is not of exactly the same shape as before. It may not be possible to determine this fact by measurement, but it may be made apparent by the fit of two such pieces together before and after one of them has had its strains changed by the partial heating.

Spontaneous cracks in cold soft steel are still not wholly explained. When such cracks occur in hard and hardened steels or in cast-irons, which have practically no ductility, they are easier to understand, though even in such cases just what was

the "last straw" which caused the rupture is usually not apparent. This is on the assumption that the composition of the steel is suitable, particularly as regards phosphorus and oxygen, which must not be present in too great proportions, and further, that the steel was well cast and rolled or forged.

Whether or not in the case of the soft-steel boiler-plate it was an additional strain due to a slight change in temperature, or vibration from some outside cause, or that the fatigue of the metal under strain progressed more rapidly than the relief of strain due to time and the operation of what might be called annealing by natural causes, is not clear. It is probably directly due to the last of these possible causes following a lessening of the ductility of the metal through working of the piece at the well-known critical temperature termed the "blue-heat." Spontaneous rupture never occurs in an old article of iron or steel, provided, of course, that no new strains have been set up within it.

2. The shape of the piece of metal affects strains very greatly when it is heated or cooled. If it be made up of relatively thick and thin parts continuously connected to each other, such as the thin web and thick hub and rim of a carwheel, on changing its temperature strains will be set up within it, not only between the interior and surface portions but between the thick and thin parts as a whole. In the case cited of a cast carwheel, these latter strains are those which are dangerous, causing the breakage and consequent destruction of the usefulness of the wheel if they be allowed to develop to their fullest extent by allowing the wheel to follow its own rate of cooling in the open. So, for a given change in temperature, the shape of the piece is very important in determining the degree of strains it possesses.

The reason for strains due to shape is that, when heated in a hotter environment or cooled in the open or in a cooler medium, the thinner parts reach the surrounding temperature sooner than the thicker parts, and consequently do their expanding or contracting so much the sooner, and strains result from the slower and later expansion or contraction of the thicker parts.

Large plane surfaces tend to intensify strains due to change of temperature of a piece of iron or steel either from heating,

cooling from a high or casting temperature, or quenching. The reason is obvious. The plane surface, cooling or heating more quickly at its edges than elsewhere, cannot yield to internal tension- or compression-stresses by change of shape, as such stresses act in straight lines within the piece. Relief may sometimes be afforded by curving or corrugating the surface, as in a casting; but a flat sheet like a saw-blade may not admit of this, and then means must be employed to conduct the operation of hardening the teeth, when the strains are set up, so that they will not crack the saw. In the case of a large circular saw-blade, partial heating, as described above, is used, and is effected by protecting the central portion of the sheet with thicker iron plates during the heating and cooling operations. The protected portion is not hardened, and has therefore a lower elastic limit and higher ductility than the hardened rim, and will thus yield a little without breaking.

3. The bulk of the piece of metal has great influence on its strains when its temperature is changed because of the distance which heat must be conducted between the surface and interior parts. When the bulk and therefore this distance is relatively great, the strains are great in proportion, and when the bulk is small they are less.

High conductivity of heat tends to offset great bulk in the strains resulting from a change in temperature.

With a piece of metal of a given contour the strains from either cause may be much reduced by reducing the bulk by means of recesses or slots or other holes properly made in the piece. The quenching operation in particular then causes much less strain.

In the case of a large tap this principle is applied by drilling a hole along its axis, which has a double effect. First, it allows the cooling fluid to cool the interior nearly as quickly as the exterior as a whole, and secondly, it reduces the actual thickness of the metal to a fraction, perhaps a third, of what it would be otherwise, and in that way reduces the strains due to bulk.

There are many ways in which the use of holes and recesses within the contour of the article may so reduce the bulk that without detriment to its use the thickness may be brought within safe limits to admit of it being cooled from a high heat

in a cooling-bath without being in danger of cracking from the internal strains set up.

Solid steel armor-plates are sometimes cracked internally by the cooling-strains arising from the quenching operation, though the steel is rather soft, with carbon from 0.20 to 0.25 per cent., except the hard face.

This may be due to either or both of two causes. (1) When the mass of the armor-plate or other piece of iron or steel is so great that the final contraction of the interior portions when cooled may not be compensated for by a reduction of cross-section of the metal (such as occurs when a test-piece of ductile metal is pulled apart in the testing-machine), then internal cracks may occur from the metal being overstrained, regardless of the ductility of the metal. (2) The ductility of the soft steel of the body of the plate is impaired by the long-continued heating without work during the carbonization by the cementation-process of the face, which is to be hard, and for that reason will not endure internal strains as well as its composition would indicate. Holes or recesses not being admissible, though they have been recommended by some, reforging or annealing to break up the coarse crystalline structure existing in the plate is resorted to, after cementation, in an effort to avoid this defect. Nevertheless, the results are not always as desired.

It does seem, however, that careful experiment should develop a method for making a large soft-steel ingot, with a high-carbon steel layer on one side, which could be forged into an armor-plate, thus avoiding the expensive and harmful cementation-process.

When an article of iron or steel is liable to crack in service from strains, the danger is sometimes best avoided by making it in two or more pieces. This practically amounts to putting cracks in the thing, but often they may be located so as to cause no trouble. The following example will illustrate: A cast-iron bottom-plate, on the center of which was cast a large ingot of steel, was broken the first time used, by the expansion of the central part heated by the molten steel. On making a plate for the purpose, of four pieces bolted together, no further trouble from this cause resulted.

4. The elastic limit of the metal affects the intensity of the

strains, because when it is low it may allow the metal to yield under the stresses which cause strains so as to relieve them in part.

5. Ductility of iron or steel sometimes allows it to yield without fracture under stresses of sufficient magnitude, so that the strains are in some degree lessened. A highly ductile metal rarely or never breaks from internal strains unless extremely massive.

A metal supposed to have great ductility may actually have very little, due perhaps to ill-treatment. The boiler-plate mentioned had probably been damaged by working at the black or blue heat, so that its ductility was impaired, and the strains arising from irregular heating and cooling were more than the remaining ductility could allow for, and rupture ensued. The armor-plates cited also had their ductility much reduced by the long heating during cementation.

When, as in cast-iron and hardened high-carbon steel, there is no ductility, and the elastic limit coincides with the ultimate strength, the strains set up in cooling cannot be relieved, and may easily reach an intensity which will cause the piece to crack or break.

6. The coefficient of expansion, strictly speaking, wholly determines the intensity of the strains arising from irregular rates of change of temperature in metals, for if it became zero such strains would not occur. The greater the rate of expansion the more intense become the strains arising from a given rate of change in temperature.

Heating a piece of iron or steel to any degree below the temperature at which it softens, will increase the strains within it in proportion to the rate of heating. Therefore, heating to relieve dangerous strain, which is often resorted to, must be very slowly done, so that the interior is heated nearly as fast as the exterior, or the extra strain due to the more rapid expansion of the exterior parts may cause the rupture it is the purpose of the heating to avoid.

By such slow heating something like the seasoning effect of time and slightly higher than atmospheric temperature is given to the piece. This effect may be used on large, massive, heat-treated articles of high-carbon steel, such as heavy cutters and projectiles and heavy articles of toughened manganese steel,

all of which are in a high state of strain, due to immersion at high temperature in a cooling-bath.

The increase of strain due to even slow and moderate heating, as, for instance, to the temperature of boiling water, may be used for testing hard heat-treated objects, such as armor-piercing projectiles, to determine whether or not they are in a dangerous condition, or rather to separate those in danger of spontaneous rupture from those which are not. Without some means for discriminating between them, some of them may lose their points by spontaneous rupture months after they are made and have traveled far.

If the article is, because of internal strain, close to or at the danger-point, the extra strain will cause it to crack or break, while, if it does not break from such heating, it will have when again cold a margin of safety represented by the additional strain caused by the gentle heating. Moreover, it has had its strains somewhat reduced thereby, and is therefore not likely afterward to break spontaneously.

When a cold massive piece of steel, as an ingot, is to be heated in a furnace, it is a matter of importance to know whether its previous cooling was rapid or slow. If rapid, it may contain so much strain as to be ruptured internally by the heating operation, while if it has been slowly cooled it may, because of smaller internal strains, endure that operation safely. To reduce strains in massive or hard steel ingots which are to be cooled to the atmospheric temperature, the rate of cooling should be retarded.

III. Internal Strains Caused by Cold-Working.

The strains due to cold-working iron or steel are the result of molecular displacement and very likely of mere changes in the inter-molecular spaces, but the theory of the matter we may for the present leave out of consideration.

The phenomena occurring with these strains include higher tensile strength, higher elastic limit, greater hardness, less ductility and lower specific gravity. Perhaps it is not claiming too much to say that these modified properties are the direct result of the strains, as if the strains be destroyed by heating to redness all these modifications disappear with them, and the properties of the metal are substantially those it possessed be-

fore the cold-working operation was performed upon it. For the purposes of this paper, therefore, these modified properties may be considered as the result of the strains.

Strains are an unavoidable accompaniment of cold-working iron and steel. Those due to cold-rolling or wire-drawing purposely applied have often useful effects when properly taken advantage of, while harmful or undesirable effects may be avoided by reducing them by annealing or other proper heat-treatment. When the pieces of metal so cold-worked are not large and the ductility is considerable there is no danger of spontaneous rupture. The cold-working may, indeed, be continued so as to cause dangerous strains if they be not relieved by heating. The service of larger pieces may be such that they are subjected to cold-working, as in the case of a die-ring for a centrifugal ore-grinding machine, which will cause flow of the metal with resulting strains, and may distort the piece or even tear it apart or break it.

Something like spontaneous rupture is met with in cold-rolled or cold-drawn shafting when the cold-working of the surface portions has so extended them that the interior is strained beyond its strength and thereby ruptured at intervals across the shaft. In that case, when the outer portions are turned off the bar may drop apart.

The beneficial effects of strains due to cold-working come from the higher tensile strength and elastic limit of the metal.

Drawn or cold-rolled shafting is much stronger per unit of cross-section than before the cold-working, and wire may be made whose ultimate strength is increased several times by the strains set up in the drawing operation. Wire-drawing strains give to some varieties of spring wire the greater part of their springiness.

The increased strength of iron and steel wire due to the strains of cold-working is made useful in many ways. In the wire cables of suspension-bridges it is relied upon and figured in as part of the tensile strength of the wire.

In wire for piano-strings and for deep sea-sounding, tensile strengths of over 400,000 lb. per sq. in. have been attained, the greater part of which is due to the internal strains set up in the wire-drawing process.

The effect of time in seasoning or annealing cold-worked

iron or steel, such as wire, is probably not known, but is very likely an appreciable amount, which, in the case of bridge-wire, for example, may reach important proportions with the lapse of decades.

Strains in cold-worked iron or steel may be detrimental in a way. A piece of straight cold-drawn shafting if machined, as, for instance, in key-seating or in turning, is very liable to be not straight after the machining, as cutting away a part of the strained metal removes part of the stresses, and those remaining cause the resultant effect to be different from what it was, and the new adjustment necessary is found in a new shape of the piece. In other words, the shaft will not be straight.

The modification and removal of strains due to cold-working by heating for long or short periods of time, one or more times, to temperatures below redness, require investigation. By these temperatures is meant those all the way down to atmospheric, and especially those from 212° to 408° F., the latter being the temperature at which a faint straw color is given to steel of a certain composition. Removal of dangerous strains by subjection to heat not great enough to discolor the surface of the metal would be a good thing in many ways, if feasible. A cold-worked piece of shafting, which will break off in the lathe when the outer skin is removed, might perhaps be practically cured by such heating between the rolling- or drawing-operations.

In the specifications for wire for a great suspension-bridge, the wire as drawn must have a tensile strength of 215,000 lb. per sq. in., and after galvanizing must have a tensile strength of 200,000 lb. per sq. in. This decrease of 15,000 lb. per sq. in. is the loss of strength due to removal of strains by heating the wire for a brief period to the galvanizing temperature, about 800° F. If the heating even at that temperature were long continued, a much larger loss of tensile strength would no doubt result.

IV. Conclusions.

The seasoning effect on iron and steel of time, referred to in the foregoing, while generally admitted to exist, is but little known, and scarcely anything has been recorded, if indeed done, toward the elucidation and enunciation of the laws which govern it.

It is held to be a fact that old iron and steel articles, years old, never have excessive strain and never fail through spontaneous rupture. It may, of course, be argued that they do not fail that way when old because if they contained enough strain to break spontaneously, they would have so broken when comparatively new; or, further, that if dangerously strained the fatigue of the metal would, in a comparatively short time, allow it to break without any diminution of the strain. These arguments may be reasonable, but many believe, nevertheless, that there is a reduction of strain in iron and steel by the seasoning effect of time.

Old cast-iron cannon which had given the normal amount of service were said to have had their strength restored by years of rest. The strains in such cannon were, it is true, the result of service-stresses, but such strains in this case are analogous to those produced by cold-working, and especially as far as they are affected by annealing by time. In the case of an article of hardened steel, seasoning or annealing by time is held to relieve strains, with the result that the piece is freed from danger of spontaneous fracture or change of shape after being finished, while if unhardened its strength and ductility are increased by such seasoning. In cold-worked steel the strength will probably decrease in time, as will the elastic limit, while the ductility will increase.

It is probable that such effect as is produced by time upon a piece of iron or steel is chiefly the result of alternate heating and cooling within the limits of the daily range of temperature, but there is evidence indicating at least that seasoning is effected by time alone, whether or not the temperature of the piece is subject to change.

It is not hard to conceive of the molecular change or rearrangement necessary to the removal of strains at ordinary temperatures if it be admitted that all molecules of matter are in a state of vibration by heat at any temperature above absolute zero or $-273°$ C. The molecules naturally would (it seems evident) shake themselves into more regular formations, which would relieve each one from crowding or being crowded by its neighbors, or the molecules would, by such formation, allow the parts in tension to lengthen and those in compression to shorten the slight amount needed to relieve the

strains in part, at least—that is, when the intensity of the strains is great. With less intensity the relief of strain from such action would manifestly be lessened, and it is hardly supposable that all strain would be eliminated by seasoning to any finite extent. Either or both of these molecular adjustments would result in diminished tension and compression, and therefore of strain in the piece.

At high temperatures this shaking of molecules into regular formations results, in time, in coarse crystallization, a further step which has results of its own in reducing strength and ductility.

Consideration of this question of seasoning by time brings up the very interesting one previously alluded to, concerning which we are as much or more in the dark, which is that of annealing iron and steel at low temperatures—that is, in the first phase, at not higher than 212° F.; in the second phase, between 212° and the lowest temperature at which steel will be given the faintest straw color, about 408°; and in the third, at temperatures between this and red heat. The temperatures giving steel the tempering colors, straw, yellow, blue, etc., vary for different grades of steel, and therefore cannot be stated exactly for steels as a whole. There is much to be learned in these fields, and the second of the three mentioned seems especially to hold out hope of practical benefit to mankind from applications of its laws. We need to know the effect on each kind of steel object, in every condition of size, shape, composition and treatment, of time, temperature and fluctuations of temperature within the limits named.

The amount of experiment to determine all these points in all degrees will be very large, but in the interest of pure as well as applied science it should be undertaken.

Annealing by time may be expected to act in a different way on cold-worked steel, in which the strains may be the result of disturbed molecular arrangements, from that on hardened steel, in which the strains result in great part, if indirectly, from the chemical constitution of the steel.

Perhaps it can be determined just how much of the hardness of hardened steel is due to chemical change and how much to strain. If there is no chemical change in heating hardened high-carbon steel to temperatures not higher than the blue

color, or even higher, say to 800° F., then it would appear that any hardening or increase of tensile strength or decrease of ductility which is removed by heating, short or continued, to temperatures not above the blue, or the higher degree mentioned, is due to the strains in the piece. The properties due to chemical constitution will then be permanent in the absence of high heat, while those due to strain may fade in time. Internal strains when in moderate degree may either increase or decrease the tensile strength of steel, as shown by many tests of steel in which annealed bars sometimes show higher, but usually lower, tensile strength than the unannealed.

That strains in hardened steel are of a quite different nature from those in cold-worked steel, may be shown by the attraction of a magnet. A piece of hardened steel with and without intense strains acts very differently from a piece of wire-drawn steel with and without strains when its attraction for or by a magnet is measured.

A file in a hardened condition, which gave a pull of 12 oz. with a common horse-shoe magnet, gave, after being heated to the blue heat, a pull of 27 oz., and after being heated red hot and slowly cooled, 31 oz. The magnet gave on its keeper a net pull of 43 oz.

Tenpenny wire nails gave with the same magnet an average pull of 27 oz., while similar nails, heated to redness and slowly cooled, gave an average pull of 24 oz. In the nails the strains seemed to have no effect in reducing the amount of the pull, but, if anything, to increase it. Still, the determination was made by crude means, and cannot be accepted as doing more than indicating the effect of the cold-working on this property. The apparatus consisted of a small spring-balance, the magnet and the pieces under examination.

In the case of the file the question is more complex. If the carbon is not changed in chemical condition by heating to the blue heat, as has been maintained, then the great increase in the susceptibility to attraction by a magnet must apparently be laid to the elimination of strains. If not, then to an allotropic modification of the iron itself.

Beyond the scope of this paper is the consideration of the chemical and physical results of rapid change of temperature, as in the hardening operation, when they both affect the strains.

There is so much to be done experimentally in that field that in the absence of the experimental data too much must be left to speculation at this time to make it worth while. No solution of the problems presented by the phenomena of the hardening and tempering of steel, however, will be complete which does not deal with the strains arising, their causes, results, and proper treatment.

Neither is the mathematical treatment of the matter of strains here considered, requiring as it does examination of intricate problems involving variations in temperature, coefficients of expansion, moduli of elasticity in tension and compression and at different temperatures, physical properties, varying kinds and intensity of strain in the different parts of the piece, and perhaps other important factors. Other investigators, better able to cope therewith, may find the matter attractive. This feature of the case must be dealt with, however, or the guns will still burst.

Heat-Treatment of Steels Containing Fifty Hundredths and Eighty Hundredths Per Cent. of Carbon.

BY C. E. CORSON, LATROBE, PA.*

(London Meeting, July, 1906.)

I. Introduction.

THE experiments of which the results and significance are set forth in this paper do not by any means cover the whole subject of the heat-treatment of the material referred to, yet they constitute a contribution to that general subject, rather than to the special department of annealing.

The material here considered is acid open-hearth steel, containing from 0.50 to 0.80 per cent. of carbon, with a nearly uniform percentage of other elements. This class of steel is rapidly increasing in commercial importance. It is even rumored that the rail-specifications of some railroads will hereafter require open-hearth steel. Yet this particular grade seems to

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have received less study and description from the standpoint of the metallographist than many others.

The investigation here described was directed to the following points:

Steel containing about 0.50 *per cent. of carbon:* 1. The structure as related to the temperature, at the same rate of cooling; 2. The structure and physical properties as related to temperature at different rates of cooling; 3. The structure and physical properties as related to different finishing-temperatures and different rates of cooling; 4. The bearing of these observations upon the statement of Prof. Albert Sauveur.[1]

Steel containing about 0.75 *per cent. of carbon:* 1. Physical properties, as related to temperature at different rates of cooling; 2. Physical properties, as related to different finishing-temperatures and different rates of cooling; 3. The bearing of these observations upon the statements of Prof. H. M. Howe.[2]

The micrographs here given as illustrations are magnified to 60 diameters. The etching-solution was one of 10 per cent. nitric acid in absolute alcohol; Seed's "process plates" were used in photographing the structure; and the prints were made on glossy "Argo" paper.

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[1] "Hot work, as such, has no influence upon the structure of the metal. Indirectly, however, by retarding crystallization until a lower temperature is reached, it may influence its structure most decidedly; but the same results could be accomplished by heat-treatment alone, *i.e.*, by re-heating the unworked metal to the temperature from which the unworked piece was allowed to cool undisturbedly." *The Metallographist*, vol. ii., p.267 (under the head of "Changes of Structure Brought About by Work").

[2] "The tensile strength at first increases with the intensity of hardening, but reaches a maximum and then declines. In case of high-carbon steel a moderate rapidity of cooling may give the highest tensile strength." *Iron, Steel and Other Alloys*, p. 221. On p. 268 of the same treatise, the following figures are given, among others:

Steel No. 193: C, 0.70; Si, 0.141; Mn, 0.068; P, 0.012; S, 0.019.

Slowly Cooled After Heating to Deg. C.	Tensile Strength. Lb. Per Sq. In.	Elastic Limit Lb. Per Sq. In.	Elong. on 8-In. Per Cent.	Reduction of Area. Per Cent.
750	82,660	40,062	12.00	25.42
1,100	92,342	59,363	13.12	20.35
1,300	48,921	29,247	1.12	17.22
1,400	41,327	33,082	1.25	15.10

Evidently, after passing 1,100°, the maximum had been reached.

The temperatures were determined by a Le Chatelier pyrometer.

It should be understood that the micrographs represent the results of treating bars of forged steel—in other words, that they do not show the effects of forging.

II. Acid Open-Hearth Steel Containing About 0.50 Per Cent. of Carbon.

1. *Structure as Related to Temperature at the Same Rate of Cooling.*

The composition of the steel was: C, 0.55; P, 0.045; Mn, 0.66; Si, 0.25; and S, 0.032 per cent. The steel was heated in a gas-furnace to various temperatures, and immediately cooled in the air, without being held at the temperature reached. The Ac of this steel was proved to be at 710° C.

Fig. 1 shows the structure of the original forged bar, and the succeeding figures its structure after heating to the temperatures stated and cooling in air, as follows: Fig. 2, to 925° C.; Fig. 3, to 650° C.; Fig. 4, to 720° C.; Fig. 5, to 750° C.; Fig. 6, to 800° C.; Fig. 7, to 850° C.; Fig. 8, to 900° C., then cooled to 550° C., re-heated to 720° C., and then cooled in air, as before. The structure shown in Fig. 2 may serve as a standard, by which the effects produced by various heat-treatments can be judged.

In these micrographs, the black areas are pearlite, Fe_3C+Fe (in the air-cooled specimens, sorbitic pearlite), while the white net-work is ferrite.

The chief point brought out by a study of this series is that, until the heating has been carried past the critical point, there is no change in structure; the previous crystallization has not been broken up. But as soon as this point has been passed in the heating-process a great change takes place, and there is a new state of crystallization. After passing through Ac, the steel has become a solid solution, and when the cooling-period begins, this point governs the size of the crystals; the higher above the point Ac, the coarser will be the structure. In this experiment, bars 9 in. long by 1.25 in. square were used, and the rate of cooling was therefore relatively rapid. Consequently, the specimens heated to the various temperatures do not show marked differences, because the rapidity in cooling tends to destroy the effect of heating to a high temperature.

Fig. 8 shows that it is only necessary to let steel cool just

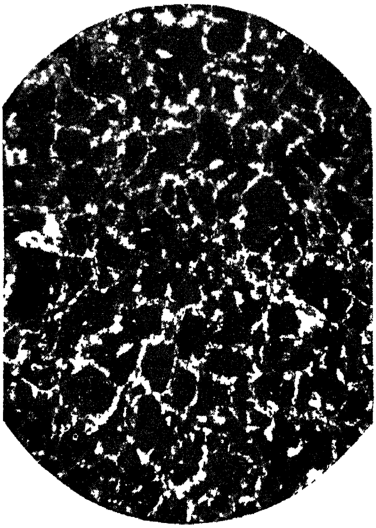


FIG. 1.
Structure of original forged bar.

FIG. 2.
Bar heated to 925° C.

FIG. 3.
Bar heated to 650° C.

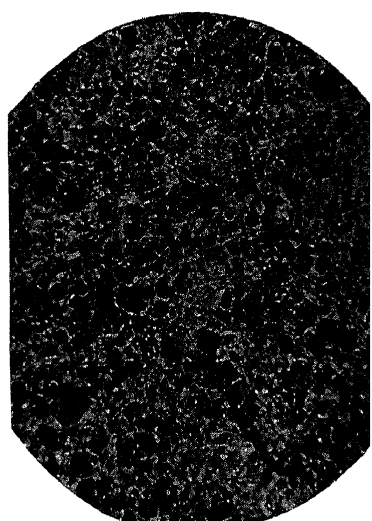


FIG. 4.
Bar heated to 720° C.

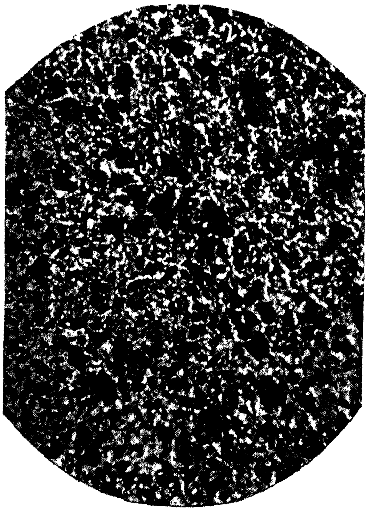


FIG. 5.
Bar heated to 750° C.

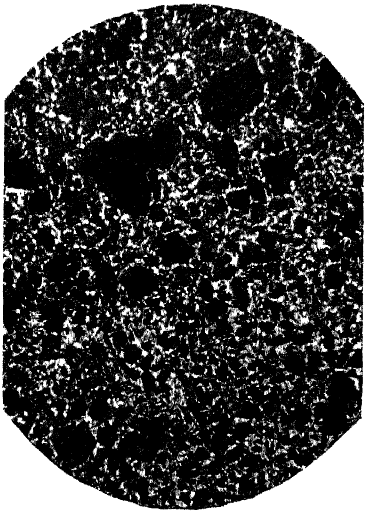


FIG. 6.
Bar heated to 800° C.

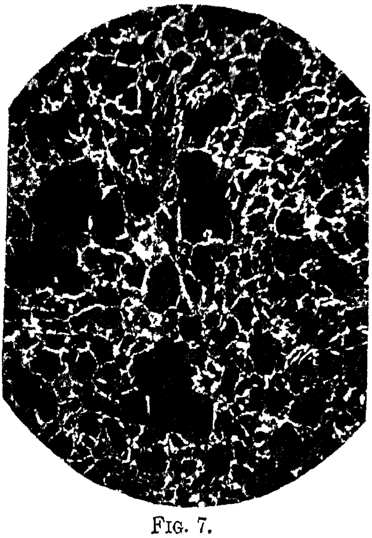


FIG. 7.
Bar heated to 850° C.

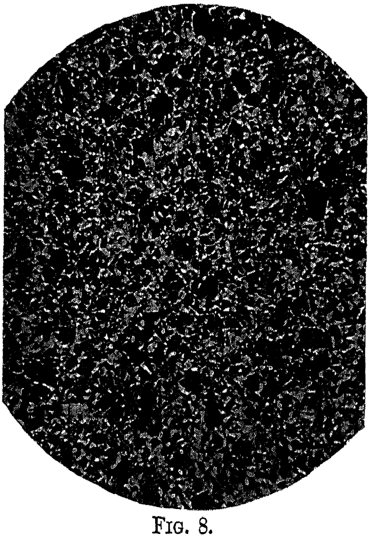


FIG. 8.
Bar heated to 900° C. and cooled to 550° C., then re-heated to 720° C. and cooled in air.

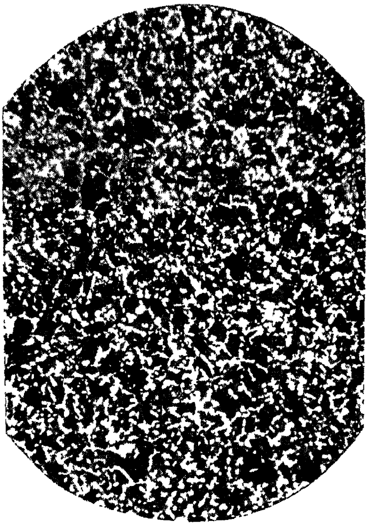


FIG. 9.
Structure of original forged bar.

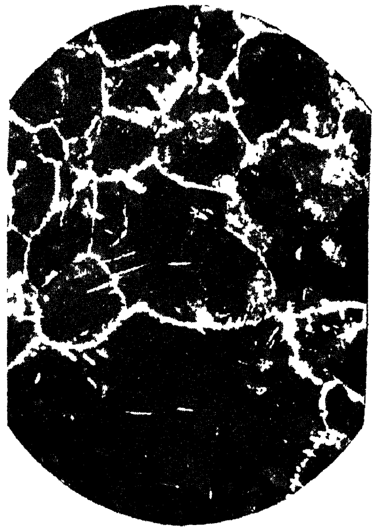


FIG. 10.
Bar heated to 800° C.; air-cooled.

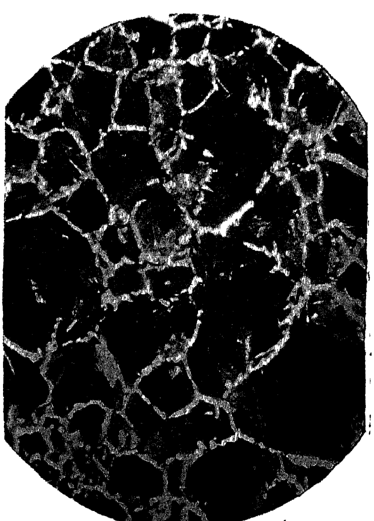


FIG. 11.
Bar heated to 440° C.; air-cooled.

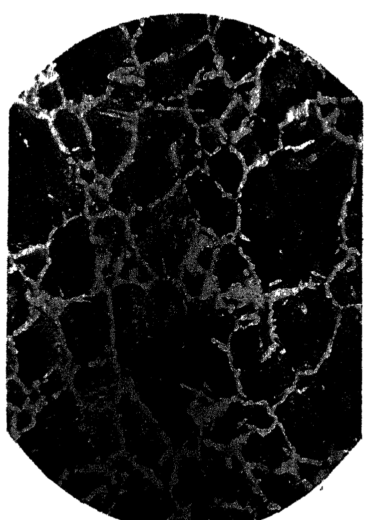


FIG. 12.
Bar heated to 660° C.; air-cooled.

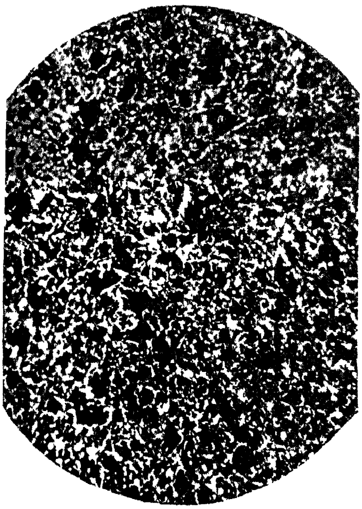


FIG. 13.
Bar heated to 710° C.; air-cooled.

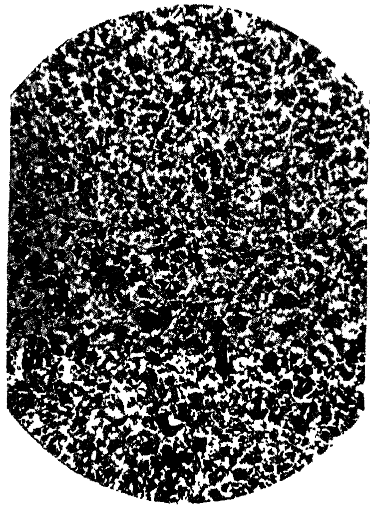


FIG. 14.
Bar heated to 710° C.; cooled in ashes.

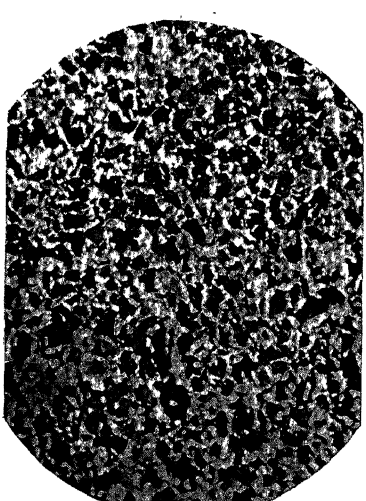


FIG. 15.
Bar heated to 710° C.; cooled in lime.

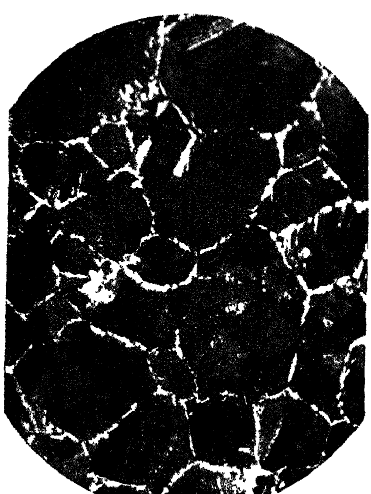


FIG. 16.
Bar heated to 780° C.; air-cooled.

FIG. 17.
Bar heated to 780° C.; cooled in ashes.

FIG. 18.
Bar heated to 780° C.; cooled in lime.

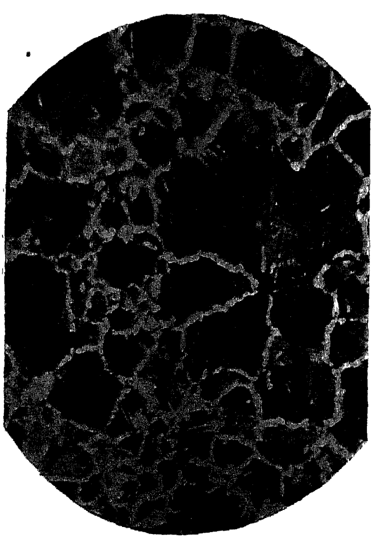


FIG. 19.
Bar heated to 900° C.; air-cooled.

FIG. 20.
Bar heated to 900° C.; cooled in ashes.

FIG. 21.
Bar heated to 900° C.; cooled in lime.

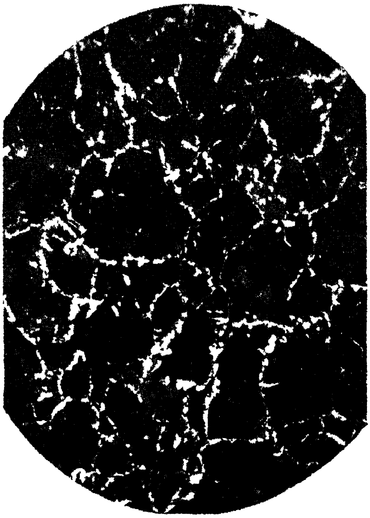


FIG. 22.
Bar hammered for 1 min. to dull yellow color; cooled in air.

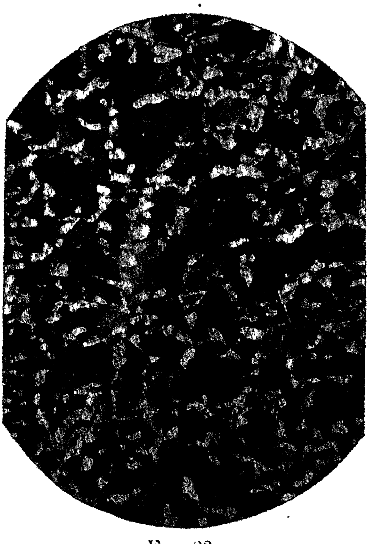


FIG. 23.
Bar hammered for 1 min. to dull yellow color; cooled in lime.

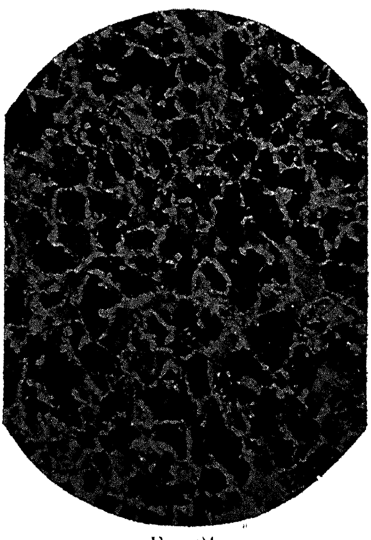


FIG. 24.
Bar hammered for 2 min. to cherry-red color; air-cooled.

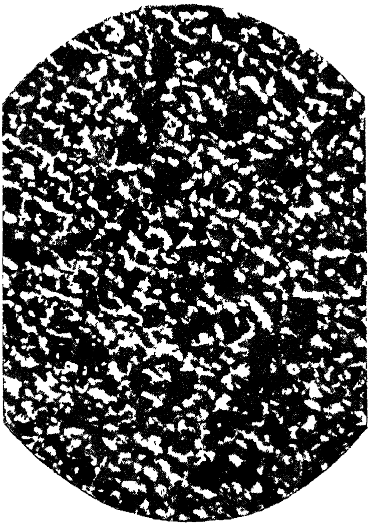


FIG. 25.
Bar hammered for 2 min. to cherry-red color; cooled in lime.

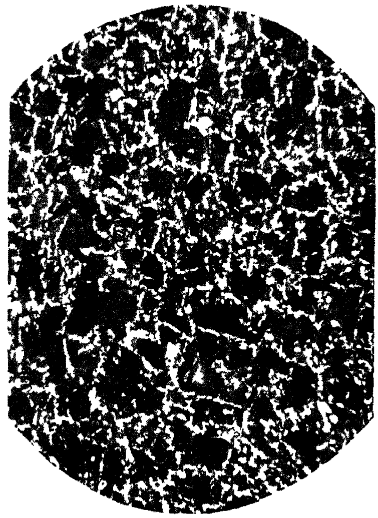


FIG. 26.
Bar hammered for 3 min. to red color; air-cooled.

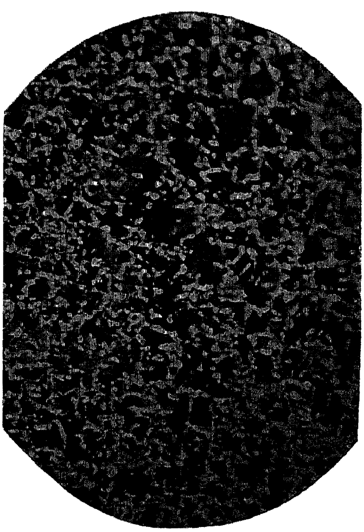


FIG. 27.
Bar hammered for 3 min. to red color; cooled in lime.

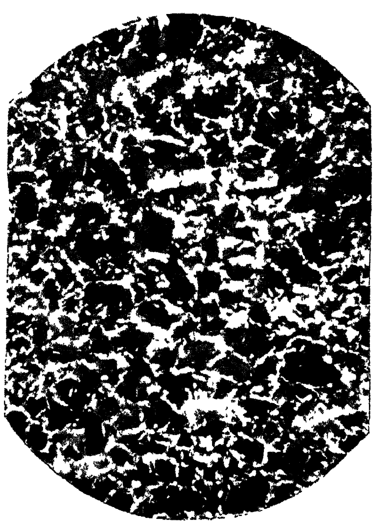


FIG. 28.
Bar hammered for 4 min. to a dark red color; air-cooled.

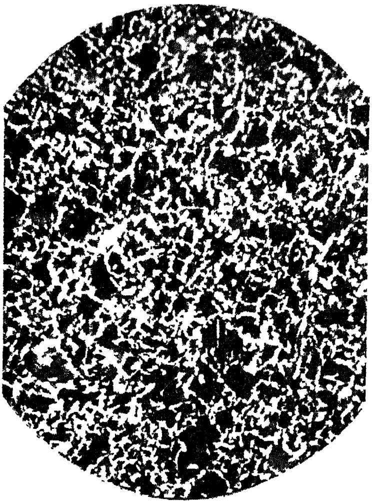


FIG. 29.
Bar hammered for 4 min. to a dark red color ; cooled in lime.

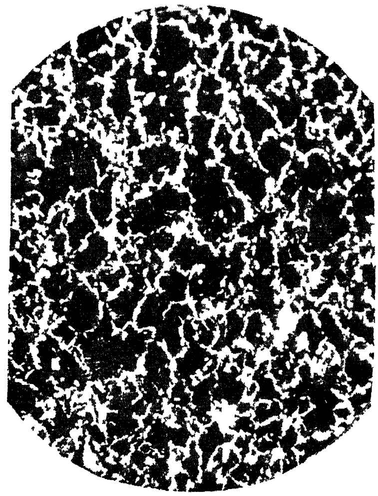


FIG. 30.
Bar hammered for about 2 min to a cherry-red (center of bar).

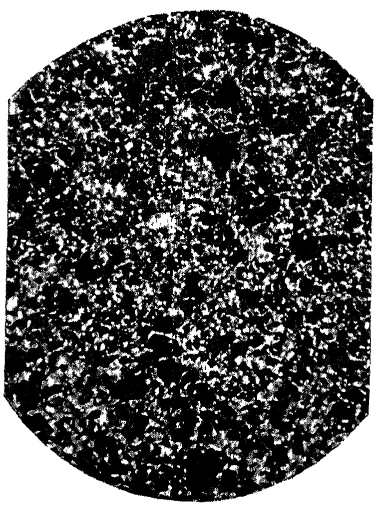


FIG. 31.
Bar hammered for about 2 min. to a cherry-red, cooled, and re-heated to match No. 30 as it left the hammer (center of bar).

below Ar in order that re-heating may produce the same results as if the steel had been allowed to cool to the temperature of the air before re-heating.

2. *Structure and Physical Tests as Related to Temperature at Different Rates of Cooling.*

The composition of this steel was: C, 0.50; P, 0.042; Mn, 0.65; Si, 0.24; S, 0.029 per cent.; and the critical point Ac was 710° C.

Figures 9 to 21, inclusive, show the structure of this steel. Figures 13 to 21, in three groups, show the effect of the different rates of cooling on the constituents of the steel. The air-cooled specimens exhibit a relatively finer structure than those cooled more slowly. The ferrite net-work is less defined because it has been absorbed by the pearlite, in the form of sorbite.

As the rate of cooling is increased by the use of ashes or lime as a medium, the ferrite separates out more completely, and has a well-defined net-work, as the pearlite becomes more a lamellar combination of $Fe_3C + Fe$.

As to the physical properties, exhibited in Table I., it will be noticed that, so long as the heating has not been carried beyond the critical point, there is no marked change in tensile strength. Beginning with Fig. 13, the groups show an interesting result as regards tensile strength and percentage of elongation. As the rate of cooling has been increased, or, in other words, as the steel has been made softer, the tensile strength has declined, while the elongation has increased. In two of the groups the percentage of contraction or reduction of area has fallen, like the tensile strength.

These illustrations indicate the effect of the development of ferrite upon the tensile strength. As the ferrite has more completely separated out and become a well-defined net-work, it has softened the steel, thereby decreasing tensile strength and increasing the stretch.

TABLE I.—*Physical Tests of the Steel, the Structure of Which is Shown in Figs. 9 to 21, Inclusive.*

Fig.	Tensile Strength. Lb. Per Sq. In.	Elongation on 1 In. Per Cent.	Reduction of Area. Per Cent.	Heated to Deg. C.	Cooled in.
9	104,176	23	46	As hammered.	Air.
10	105,960	20	36	800	Air.
11	105,450	22	35	440	Air.
12	101,120	23	37	660	Air.
13	99,847	25	43	710	Air.
14	97,555	26	31	710	Ashes.
15	95,770	25	29	710	Lime.
16	109,271	21	31	780	Air.
17	101,120	22	31	780	Ashes.
18	96,026	23	33	780	Lime.
19	106,979	19	33	900	Air.
20	100,866	21	33	900	Ashes.
21	97,809	23	31	900	Lime.

3. *Structure and Physical Properties as Related to Different Finishing-Temperatures and Different Rates of Cooling.*

This steel contained: C, 0.52; P, 0.034; Mn, 0.65; Si, 0.24, and S, 0.029 per cent. Bars, 12 in. long by 1.25 in. square, were drawn from test-ingots 4 in. square, which had been previously cogged under a steam-hammer, all these ingots having been heated to 1,000° C. (the temperature of the furnace) before work was begun. Since the finished bars were so small in section that cooling would be, in any case, comparatively rapid, only two rates of cooling were tried—namely, that of cooling in lime, and that of cooling in air within a lined box, which simply protected the bar from drafts of air in the shop. The temperature in this box was about 100° F.

The work was measured by time, since the smith hammered the bars under a small steam-hammer, giving about the same blows to each bar, without any knowledge of what the experimenter had in mind.

TABLE II.—*Physical Tests of the Steel, the Structure of Which is Shown in Figs. 22 to 29, Inclusive.*

Fig.	Cooled in.	Time of Work. Min.	Color.	Tensile Strength. Lb. Per Sq. In.	Elongation on 4 In. Per Cent.	Reduction of Area. Per Cent.
22	Air.	1	Dull yellow.	107,997	17.5	33
23	Lime.	1	....	95,515	18.0	31
24	Air.	2	Cherry-red.	104,940	16.0	36
25	Lime.	2	....	93,988	18.5	36
26	Air.	3	Red.	103,375	18.0	37
27	Lime.	3	....	94,498	21.0	39
28	Air.	4	Dark red.	100,865	17.5	36
29	Lime.	4	....	99,083	20.0	39

As in the preceding group, the effect of the lime-treatment, allowing a thorough separation of the ferrite from the mother metal, lowered in every case the tensile strength; so, conversely, by cooling at a quicker rate in the box, the tensile strength has been raised. The difference becomes greater as the cooling-period is lengthened by a higher finishing-temperature, as indicated by the smaller amount of work done upon the bar before cooling. The same marked decrease in tensile strength with continued work and consequent lower finishing-temperature is not exhibited by the lime-cooled specimens (Nos. 23, 25, 27 and 29). The large increase in tensile strength shown in No. 29 may possibly be due to an accidental longer exposure to air (by reason of slower handling, stamping, etc.) between the finish of the work and the immersion in lime. This would have allowed air-cooling to affect the result.

There has been a notion at some steel-works that the hotter a bar is finished, the lower will be its tensile strength. This experiment disproves that proposition, confirming, so far as it goes, the well-established theory that tensile strength increases, up to a certain point, with finishing-temperature. Beyond that point (which, however, was not passed in this experiment) it decreases.

4. *Prof. Sauveur's Statement.*

To test this statement, footnote [1], on page 389, two bars were drawn from the ingot having the same composition as those in Table II. The first was drawn in the usual way (the work occupying about 2 min.), and cooled in the air. When cold, it was re-heated in a coke-furnace to cherry-red; and at the same time the second bar was hammered from the hot ingot until it

was cherry-red (about 1,350° F.). When work ceased on this bar, the other was drawn from the furnace. Even to the experienced eye of a heater, there was no difference in appearance; both seemed to have the same temperature. They were then cooled, side by side, in a lined box. Fig. 30 shows the resulting structure of the bar finished at cherry-red, and Fig. 31 that of the bar re-heated to cherry-red. The latter is easily seen to be finer, showing that the identical outside appearance of the two bars was misleading as to their internal condition. The center of the worked bar was higher in temperature than that of the re-heated one. It is not possible to produce the same effect by work as by re-heating; for work preserves (and, if it be severe enough, even raises) the internal temperature, and subsequent cooling sets up an unequal crystallization. Under proper re-heating, on the other hand, the steel becomes a solid solution, from which crystals of approximate homogeneity and uniform size may separate.

Table III. shows a difference in physical qualities corresponding to that of the structure. The same difference is observed when larger masses of steel are similarly treated.

TABLE III.—*Physical Tests of Bars, of Which the Structure is Shown in Figs.* 30 *and* 31.

Fig.	Treatment.	Cooled in.	Tensile Strength. Lb. Per Sq. In.	Elongation on 4 In. Per Cent.	Reduction of Area. Per Cent.
30	Hammered about 2 min. to cherry-red.	Box.	109,271	15	26
31	Ditto; then cooled and reheated to match No. 30 as it left the hammer.	Box.	106,271	15	33

Prof. Sauveur's statement may be approximately correct for a very small section; but as the metallic mass treated is increased, the greater becomes the discrepancy of internal conditions, which invalidates his proposition.

III. ACID OPEN-HEARTH STEEL CONTAINING ABOUT 0.75 PER CENT. OF CARBON.

In the report of this part of the series of experiments, the structure will not be shown by micrographs. Such illustrations of the structure of an "æolic" or "eutectic" steel are not easily interpreted at a glance, and would in this instance add

nothing important to the evidence of the physical tests. We may, however, assume that the structural changes are analogous to those produced by the same treatment in 0.50-carbon steel; and, for the purpose of comparison, the numbers of the corresponding sections of that material are given in parentheses in the tables below.

1. *Physical Properties as Related to Temperature at Different Rates of Cooling.*

The composition of this steel was: C, 0.72; P, 0.034; Mn, 0.64; Si, 0.22; S, 0.03

TABLE IV.—*Physical Tests of Bars Nos. 32 to 44, Inclusive.*

Bar No.	Tensile Strength. Lb. Per Sq. In.	Elongation on 4 In. Per Cent	Reduction of Area. Per Cent.	Heated to Deg. C.	Cooled in.
32 (9)	143,657	13	21	As hammered.	Air.
33 (10)	139,073	11	15	300	Air.
34 (11)	124,554	13	19	440	Air.
35 (12)	136,526	11	15	660	Air.
36 (13)	129,139	9	23	710	Air.
37 (14)	123,535	14	21	710	Ashes.
38 (15)	113,092	15	19	710	Lime.
39 (16)	131,176	12	17	780	Air.
40 (17)	131,686	9	18	780	Ashes.
41 (18)	126,846	11	15	780	Lime.
42 (19)	129,393	7	23	900	Air.
43 (20)	117,677	11	15	900	Ashes.
44 (21)	119,205	9	10	900	Lime.

These results do not give as regular or as conclusive figures as those obtained from the 0.50-carbon steel; but it appears upon careful study of them that as the rate of cooling increases, the tensile strength and the contraction decrease, while there is a slight rise in elongation.

Although, as already explained, the experiments were made upon bars 1.25 in. square, and therefore can serve only as indications of what would take place in larger masses, it has been found in practical experience that these indications are highly trustworthy, and that the behavior of large masses of steel under similar conditions follows the same law.

2. *Physical Properties as Related to Different Finishing Temperatures and Different Rates of Cooling.*

This steel had the same composition as that of Table IV.

TABLE V.—*Physical Tests of Bars Nos. 45 to 56, Inclusive.*

Bar No.	Treatment. Drawn to.	Time. Min.	Cooled in.	Tensile Strength. Lb. Per Sq In.	Elongation on 4 In. Per Cent	Reduction of Area. Per Cent.
45	Dark red.	4.5	Air.	131,431	11.0	19
46	Dark red.	4.5	Box.	130,667	11.0	19
47	Dark red.	4.5	Ashes.	131,176	10.0	23
48	Red.	3	Air.	133,978	11.0	23
49	Red.	3	Box.	131,940	11.0	23
50	Red.	3	Ashes.	125,837	12.0	23
51	Cherry-red.	2	Air.	139,837	9.0	19
52	Cherry-red.	2	Box.	136,271	10.0	17
53	Cherry-red.	2	Ashes.	128,371	11.5	19
54	Dull yellow.	1	Air.	139,327	8.5	19
55	Dull yellow.	1	Box.	138,383	10.5	19
56	Dull yellow.	1	Ashes.	Broke in machine.		

3. *Prof. Howe's Statement.*

This statement, quoted in footnote [2], on page 389, is confirmed by our experiments. Table V. clearly shows an increase of tensile strength with increase of finishing-temperature. (Compare Nos. 46, 49, 52 and 55.) Table II. gives similar evidence. Tables VI. and VII. furnish further direct corroboration.

TABLE VI.—*Physical Tests of Steel Containing:* C, 0.76 ; P, 0.031; Mn, 0.64; Si, 0.26; S, ——.

Bar No.	Treatment. Drawn to.	Time. Min.	Cooled in.	Tensile Strength. Lb. Per Sq. In.	Elongation on 4 In. Per Cent.	Reduction of Area. Per Cent.
57	Dark red.	4.5	Box.	139,073	10	15
58	Red.	3.0	Box.	141,619	10	14
59	Cherry-red.	2.0	Box.	142,638	9	19
60	Dull yellow.	1.0	Box.	142,929	9	15

TABLE VII.—*Physical Tests of Steel Containing:* C, 0.74 ; P, 0.033; Mn, 0.66; Si, 0.28; S, ——.

Bar No.	Treatment. Drawn to.	Time. Min.	Cooled in	Tensile Strength. Lb. Per Sq. In.	Elongation on 4 In. Per Cent.	Reduction of Area. Per Cent.
61	Dull yellow.	1	Box.	137,544	9	17
62	Cherry-red.	2	Box.	136,526	10	17
63	Red.	3	Box.	136,271	11	19

IV. CONCLUSIONS.

These experiments have shown that, with one exception, the theories based by investigators of acknowledged authority upon experiments with other grades of steel hold good as to the two

classes of acid open-hearth steel here considered. Indeed, with regard to the one exception—namely, the qualified contradiction of Prof. Sauveur's statement as to the relative effect upon structure of mechanical work and heat-treatment—it may fairly be claimed that while the observations adduced as evidence were made upon a certain kind of steel only, the explanation which they demand is equally applicable to all other kinds, so far as their structure is affected by work or by heat-treatment. In this particular, the experiments simply declare what we would naturally expect, as an inference from the physical circumstances.

It may therefore be confessed that these experiments have done no more than bring certain commercial steels into line with others, more thoroughly investigated heretofore, in support of generally accepted theories as to heat-treatment. Yet, on the other hand, it may be asserted that such a confirmation is by no means wholly superfluous or useless. Even theorists welcome direct evidence in support of generalizations based upon analogies, or partial indications; and practitioners are prone to think (not always without some justification) that the special problems with which they have to deal are not to be solved by formulas calculated from different data. It is no small thing to convince a practical mill-man of a really general law, which governs his mill, as well as others.

I may add that, in actual practice, the results of the above experiments, and others like them, have proved highly useful as checks upon daily operations, guides to needed improvements in manipulation, and suggestions of remedy for difficulties encountered. It is not only the practical experts who may profitably realize the value of scientific inquiries and their results. The proprietors of works (including the directors and stockholders of manufacturing corporations) might well appreciate more highly the advantage of such inquiries, made by their own employees and at their own expense. Everybody has arrived (though only in comparatively recent years) at the recognition of the indispensable value of a chemical laboratory in connection with every iron-works. It cannot be said that the science of metallography has yet achieved a similar recognition. But the time is not far off when this science also will be welcomed as an aid by an industry which unquestionably needs all the scientific help it can possibly get.

Effect of Low Temperature on the Recovery of Steel From Overstrain.

BY E. J. McCAUSTLAND, ITHACA, N. Y.

(London Meeting, July, 1906.)

THE behavior of steel after overstrain and at moderate temperatures is fairly well known. It has been made the subject of much investigation, and our knowledge is clear and definite on many points. The ultimate raising of the elastic limit, after moderate overstrain, and the consequent lowering of the ductility, seem to be well established. It is also well understood that the immediate effect of overstrain is to lower the elastic limit, possibly to zero, but that if not interfered with, this limit will subsequently rise higher than before. The "tests after moderate temperature," reported in this paper, were undertaken to verify the above statements for the particular steel under investigation, and also to determine, if possible, the total time required for complete recovery from this state of overstrain.

The behavior of overstrained steel at temperatures at or below 32° F. is not so well known. Tests made by Prof. E. G. Coker, at McGill University,[1] show that steel, overstrained and subjected to cold, will remain in a state of overstrain for an indefinite period. I am not aware of any other published results of tests along this particular line.

The effect of time and temperature on the recovery of steel from overstrain was made in 1902–3 the subject of some investigation in the laboratories of the College of Civil Engineering of Cornell University. A general plan was outlined in connection with the regular courses of instruction in testing of materials; but the press of routine work and the demands of the schedules on the time of the students were so great as to presage long delay in the completion of the work. Not much had been accomplished up to 1904–05, when Messrs. E. C.

[1] *Physical Review*, vol. xv., No. 2 (1902).

Johnson and W. F. Pond chose this subject for a thesis-investigation under my direction; and I am indebted to these gentlemen for much of the experimental data upon which this paper is based. Altogether, about 120 separate tests were made, of which only a few are here reported. As instructor, I was in close touch with the experimental work in all its stages, although the actual manipulation of the testing-apparatus was done by Messrs. Johnson and Pond. The experience gained in planning the tests of the previous two years was sufficiently extended and definite to permit the adoption of the following more satisfactory outline of procedure for the tests here reported:

This outline as planned, and with brief explanation, is given in full here for reference:

I. Scheme of Experiments.

1. *Tests After Moderate Temperatures.*—Investigate the effect of time on the recovery of steel from overstrain, the specimen being subjected to moderate temperatures only—*i. e.*, temperatures ranging from 60° to 70° F.

2. *Tests After High Temperatures.*—Investigate the accelerating effect of high temperature on the time of recovery of steel from overstrain. (The high temperature referred to throughout this paper is that of saturated steam.)

3. *Tests After Low Temperatures.*—Investigate the retarding effect of low temperatures—*i. e.*, temperatures below 32° F.—on the time of recovery of steel from overstrain.

4. *Tests After Low-High and After Low-Moderate Temperatures.*—Investigate the effect of high and also of moderate temperatures on the recovery of overstrained steel which has been subjected for a considerable period to a low temperature.

II. Materials Used in These Tests.

Two kinds of steel were used in making these tests:

1. The grade known as "extra soft," the specimen being 2 in. wide and cut from a $\frac{5}{8}$-in. plate, thus giving a sectional area of 1.25 sq. in. The pieces were sheared wider than 2 in. and then milled down to gauge.

2. A mild steel, received in the form of $\frac{5}{8}$-in. round rods, which were not turned down at all, but were tested with the "skin" on. The area of these rods was taken as 0.312 sq. in.

III. Methods of Testing.

The general method followed in making these tests was the same for all the pieces tested. Measurements were taken to determine elongations on a gauge-length of 8 in., the extensometer apparatus being of the double micrometer screw type with electric contact, and reading directly to 0.001 in., and by estimation to 0.0001 in. This form of micrometer allows readings to be taken on opposite sides of the specimen, so that the mean of the readings gives the true relative deformations of the piece. To determine the stress-deformation relations, loads were applied, increasing by equal increments, and the micrometers were read at each loading, to determine the elongations. The load was occasionally reduced to zero, and the micrometers were read in order to determine the amount, if any, of the permanent elongation of the piece. In the accompanying tables, the loads are reduced to pounds per sq. in.; and the elongations determined from the micrometer-readings are reduced to inches per inch of length of specimen.

Cyclical Loadings.—When it was found that the elastic limit had been slightly exceeded, and a permanent stretch of from 0.03 to 0.06 in. in the 8-in. gauge-length had taken place, the load was entirely removed; the micrometers were reset to 8-in. gauge; and a second test was made with loads rising by equal increments up to a point slightly beyond the original elastic limit, and then returning by equal decrements to zero. So long as the specimen was in a state of overstrain, the plotting of the elongations for these cyclical loadings would form a so-called "hysteresis-loop," caused by the failure of the elastic forces within the piece to assert themselves promptly upon the removal of the external load. The width of these loops, under repeated loadings, would be reduced in proportion to the recovery from the state of overstrain. After the first cyclical loading, the bars were subjected to the temperature-conditions described for the separate tests, and the cyclical loadings were repeated at varying intervals in order to follow the process of recovery, or to demonstrate its acceleration or its complete arrest.

IV. Temperature Conditions.

The "moderate" temperatures referred to throughout this paper are those usually prevailing in a room adjoining the

laboratory, which varied from 60° to 70° F. The "high" temperatures refer to those cases in which the specimen was subjected for 45 min. to a steam-bath in a copper boiler. The "low" temperatures were obtained by placing the specimens in one of the refrigerating-rooms near the laboratory. A record of the temperatures prevailing in this room throughout the period of these tests, shows that at no time did the thermometer register above 30° F. For purposes of testing, the specimens were transported to and from the cold-storage room in a small box packed with ice. During the tests, the temperature of the laboratory was kept as low as possible, by closing the doors and opening the windows to the outer air. Since all the tests after "low" temperatures were made during the winter, the highest temperature reached in the laboratory during these tests was 41° F. The specimens were kept packed in ice, except the ends, which were left free, until wedged in the testing-machine, the pack being removed just previously to the attachment of the micrometers. A cyclical test could ordinarily be run in about 30 min., so that the effect of any small rise in temperature during the test was of no practical importance.

V. Explanation of Tabulations and Curves.

In arranging the tables which follow, both for the stress-deformation tests and for the results of the cyclical loadings, an attempt has been made to condense the figures to the lowest possible limit consistent with completeness. The original record, or log of tests, included the total applied load and the two separate micrometer-readings. In these tables the loads have been reduced to pounds per sq. in., and only the mean of the simultaneous micrometer-readings is given. From these mean values the unit-elongations, included in the tables, have been computed, which indicate the amount of stretch, per in. of length, for the recorded load in pounds per sq. in. The manner in which the curves are drawn from the tabular values is obvious. The elastic limit of a specimen is determined from an inspection of the stress-deformation diagram. The total permanent stretch taking place in the gauge-length of 8 in. was determined for each specimen by a complete release of the load and an observation of the micrometer-readings. In tab-

ulating the results of the cyclical loadings, a single column for the applied loads is made to serve for a number of tests, in order to save space on the printed page.

VI. Discussion of Tests by Groups.

1. *Moderate Temperatures.*—About 40 tests were made on eight different specimens kept at moderate temperatures, and of these specimens, two of each kind of steel have been chosen as representing the typical behavior for these conditions. Specimens Nos. 1 and 2, although of the same kind of steel, differ somewhat in behavior. The ratio of deformation to stress above a load of 33,000 lb. is considerably greater in Specimen No. 2, notwithstanding the fact that it has a slightly higher elastic limit than Specimen No. 1. On account of this fact, the cyclical loadings were not run up to such high values for No. 2 as for No. 1. While the tests on Specimen No. 1 did not extend beyond the 69-hr. period, it is to be noticed that although the recovery was practically complete, yet a slight difference in the elongations on the return-load may still be observed.

The curves for Specimens Nos. 3 and 4, of mild steel, agree more closely, notwithstanding the fact that the time of recovery is somewhat widely different. It is probable that the total stretch of No. 3, being considerably greater than that of No. 4, would operate to retard the time of complete recovery. A number of cyclical loadings, not included in the tables, were run on both these specimens. A comparison of the average times of recovery for the two kinds of steel shows that the mild steel recovers much more slowly than does the soft steel.

2. *High Temperatures.*—In this group, seven specimens were tested, a large number of observations being taken which are not here included. Specimens Nos. 5 and 6 were of the soft steel, and the sets of curves are very much alike. Attention is drawn to the differences in the form of loop No. 3 for the two specimens. In Specimen No. 5 the cycle marked No. 3 was taken immediately after the removal of the piece from the steam-bath, the temperature having been so reduced that the piece could be easily handled. In Specimen No. 6 the corresponding cycle was delayed for about an hour after the removal from the bath; and this delay resulted in the formation of a much narrower loop from the test.

Only two specimens of mild steel, Nos. 7 and 8, were tested for the effect of high temperature, and the results for both are given. It is very probable that the wide difference in the time of recovery of these two specimens was, in large measure, due to the difference in the maximum load attained in the respective cycles, which, although it amounted to only 700 lb. per sq. in., would have a comparatively large effect in the region of the elastic limit. The excess of permanent elongation in the case of Specimen No. 8 can scarcely be held accountable for any large part of this increase in the time required for complete recovery.

3. *Low Temperatures.*—In this group also, seven separate pieces were tested, and the results on four of the specimens are here given. The stress-deformation relations in Specimen No. 9, a piece of soft steel, were determined in the usual manner, and the test was followed at once by the cyclical loading, the curve for which is plotted as No. 2 on the diagram for this specimen. The piece was removed from the machine, packed in ice, and placed in cold storage, from which it was removed for a cyclical loading-test at intervals covering a period of 2,111 hours. Curve No. 5 indicates the condition of the piece after it had been at a freezing-temperature for this number of hours, and, in addition, had rested at a moderate temperature for a period of 24 hr. There is no indication whatever of recovery in this time. On the contrary, there seems to be a gradual widening of the loop throughout the whole period, which would indicate a gradual breaking-down of the material. In all, 13 cycles were run on this piece, only seven of which are plotted on the diagram. Cycles plotted as curves Nos. 6, 7 and 8 were chosen at such time-intervals as would best show the slow but fairly uniform recovery of the material from a state of overstrain.

Specimen No. 10, a piece of soft steel, was tested in much the same manner as No. 9, but was left for a much longer period under the influence of low temperature. The piece was originally tested in April and remained in cold storage all summer, the final tests being made in November. Immediately after cycle No. 5 was taken, 4,728 hr. after the original test, the piece was placed in the laboratory at moderate temperature, and cycles were run at varying intervals of time to follow

the process of recovery. Cycle No. 6, taken 120 hr. later, indicates an almost complete recovery of the elastic properties with a marked rise of the elastic limit. The figures from which cycle No. 6 was plotted are not given. It is to be noted that the time of recovery at moderate temperature of these specimens of extra soft steel, is about double the time required in the cases of those pieces which had not been subjected to the influence of low temperature for any time.

Specimen No. 11, a piece of mild steel, was tested in the same manner as Nos. 9 and 10, and was then subjected to low temperature, its condition after 2,708 hr. being indicated in the plot of cycle No. 4. The piece was then placed in a steam-bath for 45 min., after which it was left at moderate temperature. Its condition at the end of 24 hr. after the steam-bath is indicated in curve No. 5; and the following curves show a gradual recovery until curve No. 8, taken 93 hr. later, when the process of recovery seems to be practically complete. This is the only piece of mild steel that was tested under these particular temperature-conditions; but the results obtained seem to be remarkably consistent among themselves.

Specimen No. 12, of soft steel, was subjected to the same treatment as No. 11, and at the end of a period of 2,111 hr., during which it had been subjected to a temperature below 32° F., it showed little or no signs of recovery from the effects of the previous overstrain. The piece was then placed in a steam-bath for 45 min., and then left at moderate temperature for further tests. At the end of 23 hr. a cyclical loading-test was made, the results of which are plotted in curve No. 5. This test showed that complete recovery had taken place. The more rapid recovery of this piece, as compared with that of the mild steel, Specimen No. 11, is very marked.

VII. Summary of Tests and Conclusions.

Table I. gives a summary of the tests, so that the time of recovery from overstrain under different temperature-conditions, and for the two kinds of steel, may be appreciated at a glance.

An examination of Table I. shows clearly that all the tests, of which Specimens Nos. 1 to 8 are types, exhibit results quite in harmony with those obtained by other experimenters along these lines. I am not aware of any tests which take into ac-

TABLE I.—*Summary of Tests.*

No.	Kind.	Elastic Limit.	Stretch in 8 Inches.	Hours of Freezing-Temp.	Time of Recovery. Moderate Temp.	Time of Recovery. After Steam Bath	Remarks.
		Lb Sq. In.	Inches.				
1	Soft.	27,500	0.0339	None.	68		
2	Soft.	28,000	0.0366	None.	69		
3	Mild.	32,000	0.0632	None.	434	....	Increased time of recovery due to greater stretch.
4	Mild.	32,000	0.0372	None.	359		
5	Soft.	28,000	0.0517	None.	....	25	
6	Soft.	28,000	0.0512	None.	....	26	
7	Mild.	30.000	0.0129	None.	....	24	
8	Mild	30,000	0.0147	None.	....	119	Increased time of recovery possibly due to higher cyclical loading.
9	Soft	27,000	0.0244	2,111	142		
10	Soft.	27,500	0.0462	4,727	120		
11	Mild.	30,000	0.0206	2,707	....	93	
12	Soft	28,500	0.0468	2,111	....	23	

count the variation in the carbon-content of the steel, as affecting the time of recovery from overstrain; but it appears from these experiments that recovery is more gradual as the carbon increases. Specimens Nos. 5 and 6, of soft steel, show in a marked degree the accelerating effect of high temperature on the time of recovery. Since only two tests, Specimens Nos. 7 and 8, were made on the mild steel, and since these show widely varying results as to time of recovery after the application of heat, it can be concluded only that high temperatures hasten recovery to a degree which remains uncertain without further tests. As to these particular pieces, it may be possible that the difference in time of recovery is due, to some extent at least, to the higher cyclical loading to which Specimen No. 8 was subjected.

With regard to the question upon which it was the object of these experiments to obtain light, the evidence appears conclusive that the effect of continued low temperature on a piece of steel which has been stretched slightly beyond the elastic limit, is to arrest completely the recovery of its elastic properties. That is, if the permanent elongation has not exceeded, say, 1 per cent., no recovery of elastic properties will take place while the specimen is kept at a temperature at or below 32° F. Maxwell,[2] in his article on "Constitution of Bodies," says: "We know that several substances, such as gutta-percha,

[2] *Encyclopædia Britannica* (9th Amer. ed.), vol. vi., p. 276.

India-rubber, etc., may be permanently stretched when cold, and yet when afterwards heated to a certain temperature they recover their original form." This appears to be true in the case of steel, provided the deformation has not been excessive. No difference can be appreciated in this regard between the two kinds of steel included in these tests. Nor does it appear that the possibility of a final recovery is at all interfered with by a long period of rest at a temperature below 32° F. In the soft steel specimens, Nos. 9 and 10, it is true that the time required for recovery after the specimen was removed from the influence of the low temperature, was considerably greater than in the case of those specimens, Nos. 1 and 2, which had never been subjected to anything but moderate temperatures. In my opinion, however, this difference in the time of recovery was due, not to previous temperature-conditions, but rather to the differences in the percentages of elongation in the various specimens. As a final remark, having for its purpose the suggestion of an explanation of the foregoing phenomena, the closing paragraph of Prof. Maxwell's article, already cited, is pertinent:

"This view of the constitution of a solid, as consisting of groups of molecules, some of which are in different circumstances from others, also helps to explain the state of the solid after a permanent deformation has been given to it. In this case some of the less stable groups have broken up and assumed new configurations, but it is quite possible that others, more stable, may still retain their original configurations, so that the form of the body is determined by the equilibrium between these two sets of groups; but if, on account of rise of temperature, increase of moisture, violent vibration, or any other cause, the breaking up of the less stable groups is facilitated, the more stable groups may again assert their sway and tend to restore the body to the shape it had before its deformation."

TABLE II.—*Stress-Deformation Tests—Extra Soft Steel.*

Section, $\frac{5}{8}$ in. by 2 in. Area, 1.25 sq. in. Measurements on 8-in. length.

	Specimen 1.			Specimen 2.		
Load, Lb. Per Sq. In.	Mean of Micrometer Readings (Inches.)	Elongation (Inches Per Inch).	Remarks.	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Remarks.
0	0 5479	0.00000		0.6505	0.00000	
3,200	0.5472	0 00009		0 6497	0.00010	
6,400	0.5461	0.00022		0.6490	0.00019	
9,600	0.5453	0.00032		0.6482	0 00029	
12,800	0.5443	0.00045		0.6474	0.00039	
16,000	0.5434	0.00057		0.6465	0.00050	
19,200	0.5424	0.00069		0.6457	0.00060	
22,400	0.5412	0.00084		0 6450	0.00070	
25,600	0.5394	0.00106	Elastic limit, 27,500 lb. per	0 6441	0 00080	Elastic limit, 28,000 lb. per
28,800	0.5372	0.00134	sq. in.	0.6419	0.00107	sq. in.
29,600	0.5365	0.00142		0.6370	0.00170	
30,400	0.5345	0.00168		0.6190	0 00394	
31,200	0.5335	0.00180				
32,000	0.5315	0.00205				
32,800	0.5212	0.00334				
33,600	0.5048	0.00540	Permanent stretch in 8 in.			Permanent stretch in 8 in.
0	0.5140	0.00424	= 8 by 0.00424 = 0.0339 in	0.6212	0.00366	= 8 by 0.00366 = 0.0293 in.

Stress-Deformation Tests—Mild Steel.

Round rods. Diam., 0.63 in. Area, 0.312 sq. in.

	Specimen 3.			Specimen 4.		
Load, Lb. Per Sq. In.	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Remarks.	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Remarks.
0	0.5595	0.00000		0.5973	0.00000	
3,200	0.5585	0.00012		0.5963	0.00012	
6,400	0.5577	0.00022		0.5952	0.00026	
9,600	0.5568	0.00034		0.5944	0.00036	
12,800	0.5560	0.00044		0.5935	0.00047	
16,000	0.5550	0.00056		0 5926	0.00059	
19,200	0.5541	0.00067		0.5917	0.00070	
22,400	0.5533	0.00077		0.5909	0.00080	
25,600	0.5526	0.00086		0.5898	0.00094	
28,800	0.5516	0.00098		0.5887	0.00107	
30,400	No read	ing.		0.5882	0.00114	
32,000	0.5507	0 00110	Elastic limit, 32,000 lb. per	0.5873	0.00125	Elastic limit, 32,000 lb. per
33,600	0.5395	0.00250	sq. in.	0.5851	0.00152	sq. in.
35,200	0.5288	0.00384		0 5825	0.00185	
36,800	0.4869	0.00907		0.5785	0.00235	
38,400	No read	ing.	Permanent stretch in 8 in.	0.5590	0.00480	Permanent stretch in 8 in.
0	0.4963	0.00790	= 8 by 0.0079 = 0.0632 in.	0.5601	0.00465	= 8 by 0.00465 = 0.0372 in.

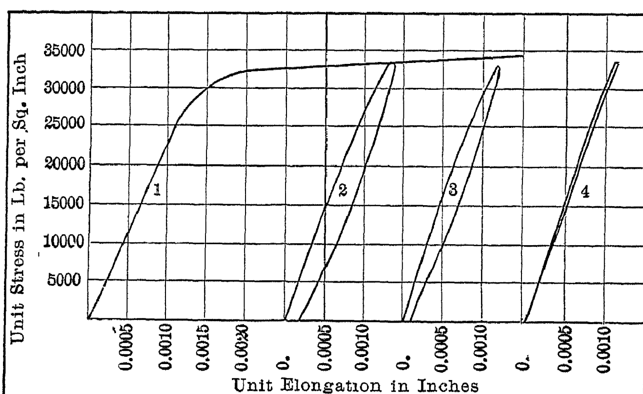
TABLE III.—*Cyclical Loading of Specimen* 1—*Extra Soft Steel.*

Section, $\frac{5}{8}$ in. by 2 in. Area, 1.25 sq. in. Measurements on 8-in. length.

Load, Lb. Per Sq. In.	15 Minutes After Overstrain.		19 Hours After Overstrain.		68 Hours After Overstrain.	
	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per (Inch).
0	0.5140	0.00000	0.5124	0.00000	0.5425	0.00000
3,200	0.5132	0.00010	0.5116	0.00010	0.5414	0.00014
6,400	0.5123	0.00021	0.5107	0.00021	0.5407	0.00022
9,600	0.5113	0.00034	0.5101	0.00029	0.5400	0.00031
12,800	0.5103	0.00046	0.5092	0.00040	0.5392	0.00041
16,000	0.5096	0.00055	0.5083	0.00051	0.5384	0.00051
19,200	0.5086	0.00068	0.5075	0.00061	0.5376	0.00061
22,400	0.5076	0.00080	0.5066	0.00072	0.5366	0.00074
25,600	0.5069	0.00090	0.5056	0.00085	0.5360	0.00081
28,800	0.5056	0.00105	0.5046	0.00097	0.5352	0.00091
32,000	0.5043	0.00121	0.5035	0.00112	0.5340	0.00106
33,200	0.5036	0.00130	0.5028	0.00120	0.5337	0.00110
32,000	0.5033	0.00134	0.5031	0.00116	0.5340	0.00106
28,800	0.5040	0.00125	0.5038	0.00107	0.5349	0.00095
25,600	0.5050	0.00112	0.5046	0.00097	0.5355	0.00087
22,400	0.5061	0.00099	0.5056	0.00085	0.5365	0.00075
19,200	0.5070	0.00088	0.5065	0.00074	0.5372	0.00066
16,000	0.5076	0.00080	0.5073	0.00064	0.5382	0.00054
12,800	0.5086	0.00068	0.5083	0.00051	0.5389	0.00045
9,600	0.5100	0.00050	0.5091	0.00041	0.5399	0.00032
6,400	0.5108	0.00040	0.5102	0.00028	0.5406	0.00023
3,200	0.5118	0.00028	0.5111	0.00016	0.5414	0.00014
0	0.5126	0.00018	0.5119	0.00006	0.5425	0.00000

Specimen 1—*Extra Soft Steel* (see Tables II. and III.).

Section, $\frac{5}{8}$ in. by 2 in. Area, 1.25 sq. in. Elastic limit, 27,500 lb. per sq. in. Permanent set in 8 in., 0.04 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.
3.—19 hr. after overstrain.
4.—68 hr. after overstrain.

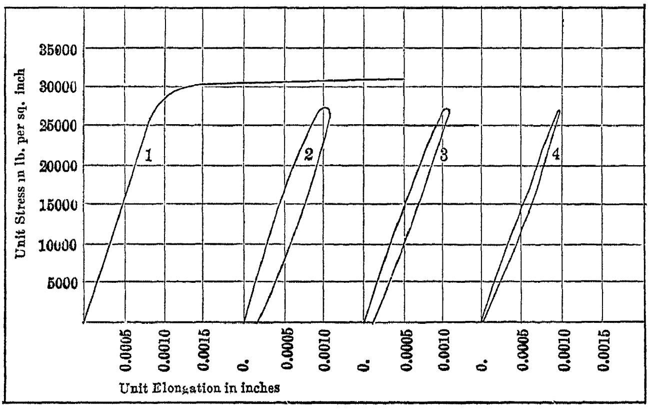
Table IV.—*Cyclical Loading of Specimen 2—Extra Soft Steel.*

Section, ⅝ in. by 2 in. Area, 1.25 sq. in. Measurements on 8 in. length.

Load, Lb Per Sq. In	15 Minutes After Overstrain.		18 Hours After Overstrain.		69 Hours After Overstrain.	
	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches per Inch).
0	0.6210	0.00000	0 6084	0 00000	0 5910	0.00000
3,200	0 6202	0 00010	0.6075	0.00011	0.5902	0.00010
6,400	0.6194	0.00020	0.6066	0.00022	0 5893	0.00021
9,600	0 6190	0.00025	0.6060	0.00030	0.5886	0.00030
12,800	0.6180	0.00038	0.6051	0.00041	0 5875	0.00044
16,000	0 6172	0.00048	0.6042	0 00052	0.5865	0.00056
19,200	0 6162	0 00060	0.6035	0.00061	0 5856	0.00067
22,400	0.6153	0.00071	0.6023	0 00076	0.5847	0.00079
25,600	0.6145	0.00081	0.6015	0.00086	0.5840	0.00088
27,200	0.6120	0 00112	0.6007	0 00097	0.5830	0.00100
25,600	0.6124	0.00107	0.6004	0.00100	0 5838	0.00090
22,400	0.6130	0.00100	0.6017	0 00084	0.5841	0.00080
19,200	0.6141	0.00086	0.6023	0 00076	0.5850	0.00075
16,000	0 6151	0.00074	0 6030	0.00067	0.5858	0.00065
12,800	0.6160	0.00062	0 6040	0.00055	0.5869	0.00051
9,600	0 6170	0.00050	0.6051	0.00041	0.5879	0.00038
6,400	0.6179	0.00039	0.6060	0.00030	0.5889	0.00026
3,200	0.6187	0.00029	0.6067	0.00021	0.5899	0.00013
0	0.6197	0.00016	0.6078	0.00008	0.5908	0.00002

Specimen 2—Extra Soft Steel (see Tables II. and IV.).

Section, ⅝ in. by 2 in. Area, 1.25 sq. in. Elastic limit, 28,000 lb. per sq. in. Permanent set in 8 in., 0.03 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.

3.—18 hr. after overstrain.

4.—69 hr. after overstrain.

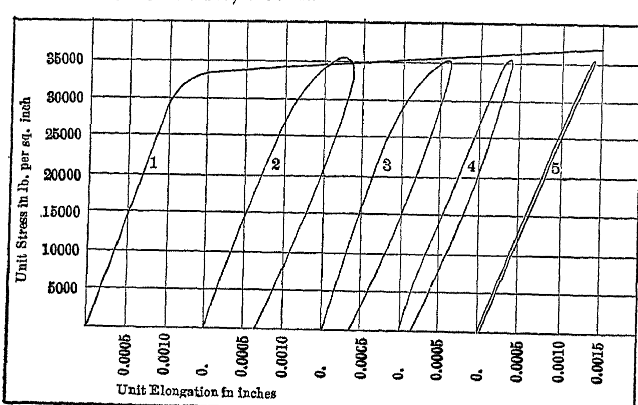
TABLE V.—*Cyclical Loading of Specimen 3—Mild Steel.*

Diam., 0.63 in. Area, 0.312 sq. in. Measurements on 8-in. length.

	15 Minutes After Overstrain.		25 Hours After Overstrain.		48 Hours After Overstrain.		434 Hours After Overstrain.	
Load, Lb. Per Sq. In.	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).
0	0.4983	0.00005	0.4913	0.00000	0.4880	0.00000	0.4190	0.00000
3,200	0.4970	0.00016	0.4904	0.00011	0.4872	0.00100	......	......
6,400	0.4962	0.00026	0.4899	0.00018	0.4863	0.00021	0.4170	0.00025
9,600	0.4955	0.00035	0.4889	0.00030	0.4854	0.00032	......	....
12,800	0.4944	0.00049	0.4880	0.00041	0.4844	0.00045	0.4150	0.00050
16,000	0.4937	0.00057	0.4871	0.00052	0.4832	0.00060	....	....
19,200	0.4925	0.00072	0.4862	0.00064	0.4826	0.00067	0.4130	0.00075
22,400	0.4915	0.00085	0.4853	0.00075	0.4815	0.00081	....	....
25,600	0.4906	0.00096	0.4842	0.00090	0.4806	0.00092	0.4110	0.00100
28,800	0.4897	0.00107	0.4832	0.00101	0.4798	0.00102	.....	....
32,000	0.4886	0.00121	0.4821	0.00115	0.4788	0.00115	0.4089	0.00126
35,200	0.4832	0.00190	0.4785	0.00160	0.4772	0.00135	0.4079	0.00139
32,000	0.4835	0.00185	0.4785	0.00160	0.4773	0.00134	0.4088	0.00127
28,800	0.4845	0.00172	0.4795	0.00147	0.4780	0.00125	....	.....
25,600	0.4852	0.00164	0.4804	0.00136	0.4785	0.00119	0.4108	0.00102
22,400	0.4862	0.00151	0.4814	0.00124	0.4799	0.00101	.....	....
19,200	0.4871	0.00140	0.4819	0.00117	0.4806	0.00092	0.4127	0.00079
16,000	0.4882	0.00126	0.4831	0.00102	0.4819	0.00076	....	......
12,800	0.4889	0.00117	0.4840	0.00091	0.4826	0.00067	0.4147	0.00054
9,600	0.4900	0.00104	0.4851	0.00077	0.4835	0.00056	....	....
6,400	0.4910	0.00091	0.4865	0.00060	0.4843	0.00046	0.4165	0.00031
3,200	0.4920	0.00079	0.4875	0.00047	0.4855	0.00031	....	.....
0	0.4928	0.00069	0.4879	0.00042	0.4862	0.00022	0.4188	0.00002

Specimen 3—Mild Steel (see Tables II. and V.).

Round rod. Diam., 0.63 in. Area, 0.312 sq. in. Elastic limit, 32,000 lb. per sq. in. Permanent set in 8 in., 0.06 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.

3.—25 hr. after overstrain.

4.—48 hr. after overstrain.

5.—434 hr. after overstrain.

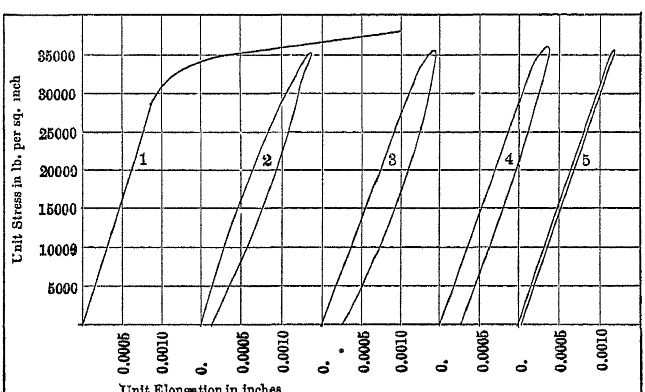
TABLE VI.—*Cyclical Loading of Specimen 4—Mild Steel.*

Diam., 0.63 in. Area, 0.312 sq. in. Measurements on 8-in. length.

	15 Minutes After Overstrain.		23 Hours After Overstrain.		47 Hours After Overstrain.		359 Hours After Overstrain.	
Load, Lb. Per Sq. In.	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).
0	0.5680	0.00000	0.5660	0.00000	0.5645	0.00000	0.5652	0.00000
3,200	0.5673	0.00009	0.5652	0.00010	0.5640	0.00006	....	....
6,400	0.5661	0.00023	0.5642	0.00022	0.5630	0.00019	0.5633	0.00024
9,600	0.5653	0.00034	0.5635	0.00031	0.5621	0.00030	....	....
12,800	0.5643	0.00046	0.5627	0.00041	0.5613	0.00040	0.5618	0.00042
16,000	0.5637	0.00054	0.5616	0.00055	0.5603	0.00052	....	....
19,200	0.5627	0.00066	0.5607	0.00066	0.5594	0.00064	0.5607	0.00056
22,400	0.5617	0.00079	0.5598	0.00077	0.5585	0.00075	....	....
25,600	0.5609	0.00089	0.5520	0.00087	0.5577	0.00085	0.5593	0.00074
28,800	0.5600	0.00100	0.5579	0.00101	0.5567	0.00097	....	....
32,000	0.5591	0.00111	0.5571	0.00111	0.5558	0.00109	0.5575	0.00097
35,200	0.5570	0.00137	0.5558	0.00128	0.5541	0.00130	0.5564	0.00110
32,000	0.5575	0.00131	0.5577	0.00129	0.5542	0.00129	0.5569	0.00103
28,800	0.5582	0.00122	0.5567	0.00116	0.5555	0.00112	....	....
25,600	0.5593	0.00109	0.5575	0.00106	0.5563	0.00102	0.5589	0.00079
22,400	0.5600	0.00100	0.5585	0.00094	0.5573	0.00090	....	....
19,200	0.5609	0.00089	0.5594	0.00082	0.5578	0.00084	0.5604	0.00060
16,000	0.5618	0.00077	0.5603	0.00071	0.5590	0.00069	....	....
12,800	0.5627	0.00066	0.5608	0.00065	0.5600	0.00056	0.5616	0.00045
9,600	0.5635	0.00056	0.5622	0.00047	0.5610	0.00044	....	....
6,400	0.5644	0.00045	0.5630	0.00037	0.5618	0.00034	0.5632	0.00025
3,200	0.5655	0.00031	0.5638	0.00027	0.5628	0.00021	....	....
0	0.5662	0.00022	0.5648	0.00015	0.5636	0.00011	0.5651	0.00001

Specimen 4—Mild Steel (see Tables II. and VI.).

Round rod. Diam., 0.63 in. Area, 0.312 sq. in. Elastic limit, 32,000 lb. per sq. in. Permanent set in 8 in., 0.04 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.
3.—23 hr. after overstrain.
4.—47 hr. after overstrain.
5.—359 hr. after overstrain.

TABLE VII.—*Stress-Deformation Tests—Extra Soft Steel.*

Section, $\frac{5}{8}$ in. by 2 in. Area, 1.25 sq. in. Measurements on 8-in. length.

Specimen 5. Specimen 6.

Load, Lb. Per Sq. In.	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Remarks.	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Remarks.
0	0 5744	0.00000		0.7079	0.00000	
3,200	0.5734	0.00012		0.7068	0.00014	
6,400	0.5728	0.00020		0.7061	0.00022	
9,600	0.5720	0.00030		0.7050	0 00036	
12,800	0.5715	0 00036		0.7043	0.00045	
16,000	0.5708	0 00045		0.7035	0 00055	
19,200	0.5701	0.00054		0.7028	0.00064	
22,400	0.5693	0.00064		0.7019	0.00075	
25,600	0.5683	[illegible]		0.7010	0 00086	
27,200	0.5673	[illegible]	[illegible] per	0.7000	0.00099	Elastic limit, 28,000 lb per
28,800	0.5670	0.00092	sq. in.	0.6995	0.00105	sq. in.
30,400	0.5663	0.00101		0.6985	0 00117	
32,000	0.5645	0 00124		0.6975	0.00130	
32,800	0.5635	0.00136		0.6970	0 00136	
33,600	0.5618	0.00157		0.6954	0 00156	
35,200	0.5495	0 00310		0.6857	0.00277	
36,000	0.5408	0.00420		0.6477	0.00752	
36,800	0.4985	0 00950	Permanent stretch in 8 in.	No read	ing.	Permanent stretch in 8 in.
0	0.5227	0.00646	[illegible]	[illegible] 6567	0 00640	=8 by 0.00640 = 0.0512 in.

TABLE VIII.—*Stress-Deformation Tests—Mild Steel.*

Round rods. Diam., 0.63 in. Area, 0.312 sq. in.

Specimen 7. Specimen 8.

Load, Lb. Per Sq. In.	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Remarks.	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Remarks.
0	0.4899	0.00000		0.4695	0.00000	
3,200	0.4889	0.00012		0.4687	0.00010	
6,400	0.4876	0.00029		0.4676	0.00024	
9,600	0.4865	0.00042		0.4669	0.00032	
12,800	0.4853	0.00057		0.4660	0.00044	
16,000	0.4845	0.00067		0.4653	0.00052	
19,200	0 4835	0.00080		0.4646	0.00061	
22,400	0.4823	0.00095		0.4639	0.00070	
25,600	0.4810	0.00111		0.4629	0.00082	
28,800	0.4804	0.00119	Elastic limit, 30,000 lb. per	0.4621	0.00092	Elastic limit, 30,000 lb. per
30,400	0.4797	0.00127	sq. in.	0.4616	0.00099	sq. in.
32,000	0 4788	0.00139		0.4609	0.00107	
33,600	0.4771	0.00160		0.4594	0.00126	
35,200	0.4682	0.00271		0.4528	[illegible]	
36,800	No read	ing	Permanent stretch in 8 in.	0.4455	[illegible]	Permanent stretch in 8 in.
0	0 4770	[illegible]	= 8 by [illegible]	0.4548	0.00184	= 8 by 0.00184=0.0147 in.

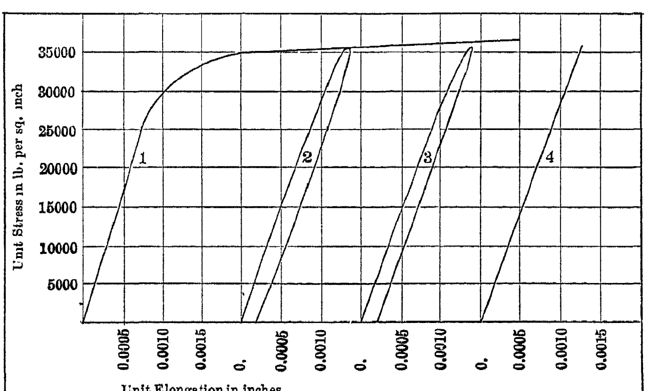
TABLE IX.—*Cyclical Loading of Specimen 5—Extra Soft Steel.*

Section, ⅝ in. by 2 in. Area, 1.25 sq. in. Measurements on 8-in. length.

Load, Lb. Per Sq. In.	15 Minutes After Overstrain.		2 Hours After Overstrain, and 15 Minutes after Steam-Bath of ¾ Hour.		25 Hours After Overstrain.	
	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).
0	0.5338	0.00000	0.5960	0.00000	0.5939	0.00000
3,200	0.5328	0.00012	0.5950	0.00013	0.5930	0.00011
6,400	0.5322	0.00020	0.5939	0.00026	0.5920	0.00024
9,600	0.5312	0.00032	0.5934	0.00033	0.5912	0.00034
12,800	0.5303	0.00044	0.5925	0.00044	0.5903	0.00045
16,000	0.5295	0.00054	0.5915	0.00056	0.5895	0.00055
19,200	0.5287	0.00064	0.5909	0.00064	0.5886	0.00066
22,400	0.5275	0.00079	0.5898	0.00078	0.5880	0.00074
25,600	0.5265	0.00091	0.5885	0.00094	0.5868	0.00089
28,800	0.5255	0.00104	0.5878	0.00103	0.5855	0.00104
32,200	0.5247	0.00114	0.5870	0.00113	0.5350	0.00111
35,000	0.5233	0.00131	0.5846	0.00142	0.5841	0.00122
32,000	0.5234	0.00130	0.5855	0.00131	0.5848	0.00114
28,800	0.5242	0.00120	0.5860	0.00125	0.5854	0.00106
25,600	0.5250	0.00110	0.5869	0.00113	0.5867	0.00090
22,400	0.5257	0.00101	0.5877	0.00103	0.5881	0.00072
19,200	0.5270	0.00085	0.5888	0.00090	0.5886	0.00066
16,000	0.5277	0.00076	0.5898	0.00077	0.5896	0.00054
12,800	0.5284	0.00067	0.5914	0.00057	0.5902	0.00046
9,600	0.5297	0.00051	0.5919	0.00051	0.5910	0.00036
6,400	0.5305	0.00041	0.5926	0.00042	0.5920	0.00024
3,200	0.5313	0.00031	0.5935	0.00031	0.5930	0.00011
0	0.5320	0.00023	0.5945	0.00019	0.5940	0.00000

Specimen 5—Extra Soft Steel (see Tables VII. and IX.).

Section, ⅝ in. by 2 in. Area, 1.25 sq. in. Elastic limit, 28,000 lb. per sq. in Permanent set in 8 in., 0.05 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.

3.—2 hr. after overstrain and 15 min. after steam-bath of 45 min.

4.—25 hr. after overstrain and 23 hr. after steam-bath.

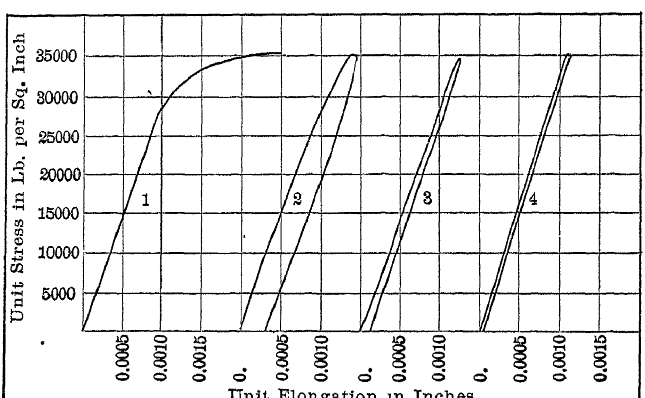
TABLE X.—*Cyclical Loading of Specimen* 6—*Extra Soft Steel.*

Section, $\frac{5}{8}$ in. by 2 in. Area, 1.25 sq. in. Measurements on 8-in. length.

Load, Lb. Per Sq. In.	15 Minutes After Overstrain.		3 Hours After Overstrain, and 1 Hour After Steam-Bath of 45 Min.		26 Hours After Overstrain.	
	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches)	Elongation (Inches Per Inch).
0	0.6574	0 00000	0.6457	0.00000	0.6453	0.00000
3,200	0 6565	0.00011	0.6446	0.00014	0 6446	0.00009
6,400	0.6553	0.00026	0.6439	0.00022	0.6436	0.00021
9,600	0.6547	0.00034	0.6430	0.00034	0.6428	0.00031
12,800	0.6539	0.00044	0.6422	0.00044	0.6421	0.00046
16,000	0.6531	0.00054	0.6412	0.00056	0.6413	0.00050
19,200	0.6520	0.00067	0.6404	0.00066	0.6405	0 00060
22,400	0.6510	0.00080	0.6397	0.00075	0.6395	0.00072
25,600	0 6502	0.00090	0.6388	0.00086	0.6388	0.00081
28,800	0.6493	0.00101	0.6378	0.00098	0.6378	0.00094
32,000	0.6480	0.00117	0.6371	0.00107	0.6372	0.00101
35,000	0.6460	0.00142	0.6361	0.00120	0.6363	0.00112
32,200	0 6460	0.00142	0.6370	0.00109	0.6373	0.00100
28,800	0 6466	0.00135	0.6373	0.00105	0.6379	0.00092
25,600	0.6478	0 00120	0.6382	0.00094	0.6389	0.00080
22,400	0.6486	0.00110	0.6390	0.00084	0.6396	0.00071
19,200	0.6494	0.00100	0.6400	0.00071	0.6405	0.00060
16,000	0.6501	0.00091	0.6407	0.00062	0.6412	0 00051
12,800	0.6512	0.00077	0.6015	0.00052	0.6422	0 00039
9,600	0.6522	0.00065	0.6423	0.00042	0 6429	0.00030
6,400	0 6531	0.00054	0.6432	0.00031	0.6437	0.00020
3,200	0.6540	0.00042	0.6440	0.00021	0.6446	0.00009
0	0.6549	0.00031	0.6447	0.00012	0.6453	0.00000

Specimen 6—*Extra Soft Steel* (see Tables VII. and X.).

Section, $\frac{5}{8}$ in. by 2 in. Area, 1.25 sq. in. Elastic limit, 28,000 lb. per sq. in. Permanent set in 8 in., 0.05 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.

3.—3 hr. after overstrain and 1 hr. after steam-bath of 45 min.

4.—26 hr. after overstrain and 24 hr. after steam-bath.

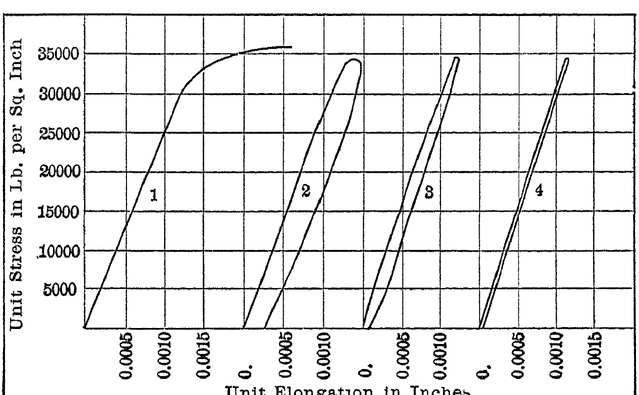
TABLE XI.—*Cyclical Loading of Specimen 7—Mild Steel.*

Round rod. Diam., 0.63 in. Area, 0.312 sq. in. Measurements on 8-in length.

Load, Lb. Per Sq. In.	15 Minutes After Overstrain.		2 Hours After Overstrain, and ½ Hour after Steam-Bath of ¾ Hour.		24 Hours After Overstrain.	
	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean o Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).
0	0.4772	0.00000	0.4481	0.00000	0 4470	0.00000
6,400	0.4755	0.00021	0.4460	0.00026	0.4452	0.00022
12,800	0.4735	0.00046	0.4447	0.00042	0.4435	0.00044
19,200	0.4721	0.00064	0.4431	0.00062	0.4416	0.00067
25,600	0.4702	0.00087	0.4418	0.00079	0.4400	0.00087
32,000	0.4685	0.00109	0.4396	0.00106	0.4385	0.00106
34,500	0.4662	0.00137	0.4383	0.00122	0.4379	0.00114
32,000	0.4659	0.00141	0.4388	0.00116	0.4384	0.00107
25,600	0.4577	0.00119	0 4407	0.00092	0.4400	0.00087
19,200	0.4695	0.00096	0 4423	0.00072	0.4415	0.00069
12,800	0 4710	0 00077	0.4441	0.00050	0.4434	0.00045
6,400	0.4726	0 00057	0.4455	0.00032	0.4450	0.00025
0	0.4748	0.00030	0 4468	0.00016	0.4470	0.00000

Specimen 7—Mild Steel (see Tables VIII. and XI.).

Round rod. Diam., 0.63 in. Area, 0.312 sq. in. Elastic limit, 30,000 lb. per sq. in. Permanent set in 8 in., 0.013 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.

3.—2 hr. after overstrain and 30 min. after steam-bath of 45 min.

4.—24 hr. after overstrain and 22 hr. after steam-bath.

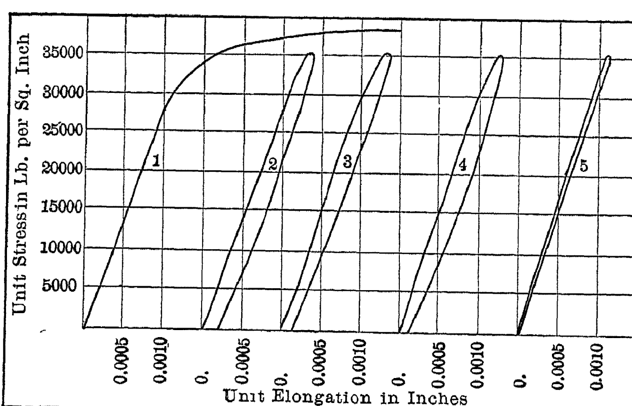
TABLE XII.—*Cyclical Loading of Specimen 8—Mild Steel.*

Round rod. Diam., 0.63 in. Area, 0.312 sq. in. Measurements on 8-in. length.

Load, Lb. Per Sq. In.	15 Minutes After Overstrain.		24 Hours After Overstrain.		49 Hours After Overstrain.		119 Hours After Overstrain.	
	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch)	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).
3,200	0.4550	0.00000	0.4520	0.00000	0.4480	0.00000	0.4430	0.00000
6,400	0.5430	0.00025	0.4497	0.00029	0.4460	0.00025	0.4411	0.00024
12,800	0.4513	0.00046	0.4482	0.00047	0.4442	0.00047	0.4392	0.00048
19,200	0.4497	0.00066	0.4464	0.00070	0.4425	0.00069	0.4377	0 00066
25,600	0.4480	0.00087	0.4449	0.00089	0.4410	0.00087	0.4360	0 00088
32,000	0.4459	0.00114	0.4430	0.00112	0.4392	0.00110	0.4344	0.00108
35,200	0.4442	0.00135	0.4407	0.00141	0.4367	0.00141	0.4332	0.00122
32,000	0.4446	0.00130	0.4408	0.00140	0.4371	0 00136	0.4342	0.00110
25,600	0.4461	0.00111	0.4425	0.00119	0.4387	0.00116	0.4358	0.00090
19,200	0.4480	0.00087	0.4442	0.00097	0 4402	0.00097	0.4376	0.00067
12,800	0.4495	0.00069	0.4457	0 00079	0.4422	0.00072	0.4390	0 00050
6,400	0.4516	0.00042	0.4475	0.00056	0.4437	0.00054	0.4409	0 00026
0	0.4536	0.00017	0.4497	0.00029	0.4459	0.00026	0.4428	0.00002

Specimen 8—*Mild Steel* (see Tables VIII. and XII.).

Round rod. Diam., 0.63 in. Area, 0.312 sq. in. Elastic limit, 30,000 lb. per sq. in. Permanent set in 8 in., 0.015 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.

3.—24 hr. after overstrain and 22 hr. after steam-bath of 45 min.

4.—49 hr. after overstrain and 47 hr. after steam-bath.

5.—119 hr. after overstrain and 117 hr. after steam-bath.

TABLE XIII.—*Cyclical Loading of Specimen 9—Extra Soft Steel.*

Section, ⅝ in. by 2 in. Area, 1.25 sq. in. Measurements on 8-in. length.

Load, Lb. Per Sq. In.	15 Minutes After Overstrain.		48 Hours After Overstrain, and 47 Hours at Freezing.		2092 Hours After Overstrain, and 2091 Hours at Freezing.		2136 Hours After Overstrain, 2112 at Freezing and 24 Hours at Moderate.	
	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).
0	0.5709	0.00000	0.5252	0.00000	0.5127	0.00000	0.4990	0.00000
3,200	0.5700	0.00011	0.5244	0.00010	0.5117	0.00012	0.4980	0.00012
6,400	0.5689	0.00025	0.5235	0.00021	0.5106	0 00026	0.4971	0.00024
9,600	0.5682	0.00034	0.5227	0.00031	0.5097	0.00037	0.4967	0.00029
12,800	0.5672	0.00046	0.5220	0.00040	0.5090	0.00046	0.4952	0.00047
16,000	0.5665	0.00055	0.5207	0.00056	0.5077	0.00062	0.4946	0.00055
19,200	0.5658	0.00064	0.5200	0.00065	0.5066	0.00076	0.4940	0.00062
22,400	0.5649	0.00075	0.5194	0.00072	0.5058	0.00086	0.4932	0.00072
25,600	0.5638	0.00089	0.5184	0.00085	0.5046	0.00101	0.4914	0.00095
28,800	0.5595	0.00142	0.5135	0.00146	0.4984	0.00179	0.4845	0.00181
25,600	0.5596	0.00141	0.5135	0.00146	0.4982	0.00181	0.4846	0.00180
22,400	0.5600	0.00136	0.5143	0.00136	0.4890	0.00171	0.4853	0.00171
19,200	0.5610	0.00124	0.5146	0.00132	0.4997	0.00162	0.4860	0.00162
16,000	0.5621	0.00110	0.5156	0.00120	0.5005	0.00152	0.4865	0.00156
12,800	0.5631	0.00097	0.5165	0.00109	0.5014	0.00141	0.4872	0.00147
9,600	0.5639	0.00087	0.5172	0.00100	0.5023	0.00130	0.4886	0.00130
6,400	0.5647	0.00077	0.5180	0.00091	0.5031	0.00120	0.4891	0.00124
3,200	0.5659	0.00062	0.5189	0.00079	0.5042	0.09106	0.4899	0.00114
0	0.5669	0.00050	0.5202	0.00062	0.5050	0.00096	0.4911	0.00098

Load, Lb. Per Sq. In.	2183 Hours After Overstrain, 2112 at Freezing and 71 at Moderate.		2231 Hours After Overstrain, 2112 at Freezing and 119 at Moderate.		2254 Hours After Overstrain, 2112 at Freezing and 142 at Moderate.	
0	0.4923	0.00000	0.6111	0.00000	0.6085	0.00000
3,200	0.4911	0.00015	0.6104	0.00009	0.6075	0.00012
6,400	0.4901	0.00028	0.6095	0.00020	0.6067	0 00022
9,600	0.4892	0.00039	0.6085	0.00033	0.6058	0.00034
12,800	0.4883	0.00050	0.6080	0.00039	0.6050	0.00044
16,000	0.4872	0.00064	0.6069	0.00052	0.6045	0.00050
19,200	0.4868	0.00069	0.6060	0.00064	0.6035	0.00062
22,400	0.4855	0.00085	0.6052	0.00074	0.6025	0.00075
25,600	0.4851	0.00090	0.6045	0.00083	0.6017	0.00085
28,800	0.4835	0.00110	0.6029	0.00102	0.6002	0.00104
25,600	0.4836	0.00109	0.6030	0.00101	0.6017	0.00085
22,400	0.4846	0.00096	0.6036	0.00094	0.6021	0.00080
19,200	0.4855	0.00085	0.6045	0.00082	0.6031	0.00067
16,000	0.4863	0.00075	0.6054	0.00071	0.6040	0.00056
12,800	0.4870	0.00066	0.6061	0.00062	0.6050	0.00044
9,600	0.4880	0.00054	0.6070	0.00051	0.6059	0.00032
6,400	0.4888	0.00044	0.6082	0.00036	0.6065	0.00025
3,200	0.4895	0.00035	0.6089	0.00028	0.6074	0.00014
0	0.4905	0.00022	0.6096	0.00019	0.6085	0.00000

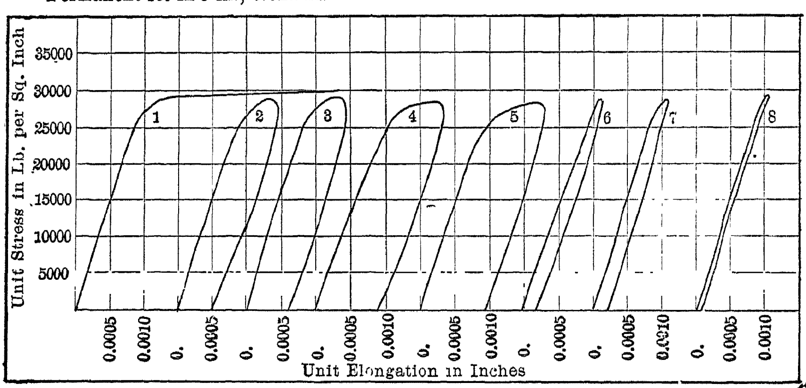
TABLE XIV.—*Stress-Deformation Tests—Extra Soft Steel.*

Section ⅝ in. by 2 in. Area, 1.25 sq. in. Measurement on 8-in. length.

Load, Lb. Per Sq. In.	Specimen 9. Mean of Micrometer Readings (Inches)	Specimen 9. Elongation (Inches Per Inch).	Specimen 9. Remarks.	Specimen 10. Mean of Micrometer Readings (Inches).	Specimen 10. Elongation (Inches Per Inch).	Specimen 10. Remarks.
0	0.5946	0.00000		0 9733	0.00000	
3,200	0.5938	0.00010		0.9729	0 00005	
6,400	0.5930	0.00020		0.9721	0.00015	
9,600	0 5921	0.00031		0 9708	0.00031	
12,800	0.5910	0.00045		0.9705	0.00035	
16,000	0.5905	0.00051		0.9693	0.00050	
19,200	0.5895	0.00064		0.9684	0.00061	
22,400	0 5882	0.00080		0 9670	0.00079	
24,000	0.5878	0.00085		0.9668	0.00085	
25,600	0.5875	[illegible]		0.9658	0.00094	
26,400	0 5870	[illegible]	[illegible] lb. per	No read	ing.	Elastic limit, 27,500 lb. per
27,200	0 5865	0.00101	sq. in.	0.9648	0.00106	sq. in.
28,000	0 5863	0.00104		No read	ing	
28,800	0.5801	0.00181		[illegible]	[illegible]	
29,600	0.5757	0.00236		[illegible]	[illegible]	
30,000	0 5642	0.00380		[illegible]	[illegible]	
32,000	No read	ing		[illegible]	[illegible]	
35,200	[illegible]	[illegible]	Permanent stretch in 8 in.	[illegible]	[illegible]	[illegible] in 8 in.
0	[illegible]	[illegible]	=8 by 0.00305=0.0244 in.	0.9271	0.00577	=8 by 0 00577=0.0462 in.

Specimen 9—Extra Soft Steel (see Tables XIII. and XIV.).

Section, ⅝ in. by 2 in. Area, 1.25 sq. in. Elastic limit, 27,000 lb. per sq. in. Permanent set in 8 in., 0.024 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.

3.—48 hr. after overstrain, 47 hr. at freezing-temperature (1 hr. interval).

4.—2,092 hr. after overstrain, 2,091 hr. at freezing-temperature.

5.—2,136 hr. after overstrain, 2,111 hr. at freezing, 24 hr. at moderate temperature.

6.—2,183 hr. after overstrain, 2,111 hr. at freezing, 71 hr. at moderate.

7.—2,231 hr. after overstrain, 2,111 hr. at freezing, 119 hr. at moderate.

8.—2,254 hr. after overstrain, 2,111 hr. at freezing, 142 hr. at moderate.

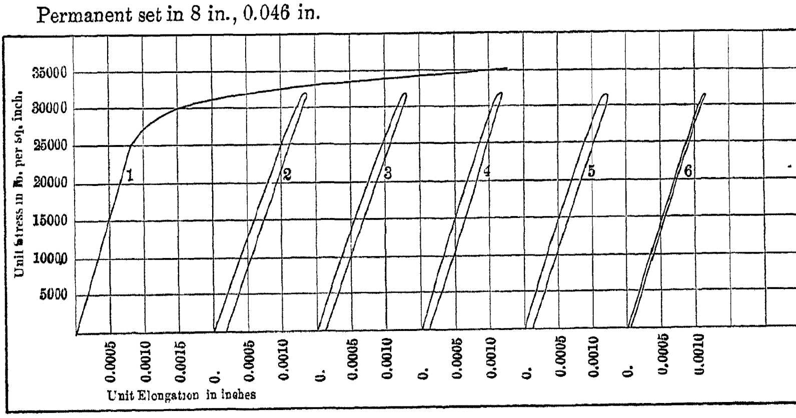
TABLE XV.—*Cyclical Loading of Specimen* 10—*Extra Soft Steel.*

Section, ⅝ in. by 2 in. Area, 1.25 sq. in. Measurements on 8-in. length.

Load, Lb. Per Sq. In.	24 Hours After Overstrain, 23 at Freezing.		168 Hours After Overstrain, 167 at Freezing.		361 Hours After Overstrain, 360 at Freezing.		4728 Hours After Overstrain, 4727 at Freezing.	
	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).
0	0.0320	0.00000	0.0165	0.00000	0.0053	0.00000	0.5968	0.00000
3,200	0.0331	0.00014	0.0178	0.00016	0.0061	0.00010	0.5958	0.00012
6,400	0.0340	0.00025	0.0185	0.00025	0.0071	0.00022	0.5947	0.00026
9,600	0.0352	0.00040	0.0196	0.00039	0.0081	0.00035	0.5937	0.00039
12,800	0.0362	0.00052	0.0206	0.00051	0.0090	0.00046	0.5928	0.00050
16,000	0.0372	0.00065	0.0215	0.00062	0.0102	0.00061	0.5919	0.00061
19,200	0.0382	0.00077	0.0228	0.00079	0.0105	0.00065	0.5909	0.00074
22,400	0.0391	0.00089	0.0233	0.00085	0.0116	0.00079	0.5901	0.00084
25,600	0.0400	0.00100	0.0243	0.00097	0.0126	0.00091	0.5892	0.00095
28,800	0.0412	0.00115	0.0254	0.00111	0.0137	0.00105	0.5884	0.00105
32,000	0.0425	0.00131	0.0270	0.00131	0.0149	0.00120	0.5870	0.00122
28,800	0.0417	0.00121	0.0258	0.00116	0.0142	0.00111	0.5879	0.00111
25,600	0.0408	0.00110	0.0248	0.00104	0.0135	0.00102	0.5887	0.00101
22,400	0.0399	0.00099	0.0238	0.00091	0.0126	0.00091	0.5895	0.00091
19,200	0.0390	0.00087	0.0234	0.00086	0.0117	0.00080	0.5902	0.00082
16,000	0.0381	0.00076	0.0223	0.00072	0.0108	0.00069	0.5914	0.00067
12,800	0.0373	0.00066	0.0213	0.00060	0.0101	0.00060	0.5921	0.00059
9,600	0.0363	0.00054	0.0204	0.00049	0.0090	0.00046	0.5927	0.00051
6,400	0.0353	0.00041	0.0194	0.00036	0.0081	0.00035	0.5940	0.00035
3,200	0.0344	0.00030	0.0185	0.00025	0.0072	0.00024	0.5949	0.00024
0	0.0335	0.00019	0.0178	0.00016	0.0064	0.00014	0.5957	0.00014

Specimen 10—*Extra Soft Steel* (see Tables XIV. and XV.).

Section, ⅝ in. by 2 in. Area, 1.25 sq. in. Elastic limit, 27,500 lb. per sq. in. Permanent set in 8 in., 0.046 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—24 hr. after overstrain, 23 hr. at freezing-temperature (1 hr. interval).

3.—168 hr. after overstrain, 167 hr. at freezing-temperature.

4.—361 hr. after overstrain, 360 hr. at freezing-temperature.

5.—4,728 hr. after overstrain, 4,727 hr. at freezing-temperature.

6.—4,848 hr. after overstrain, 4,727 hr. at freezing, 120 hr. at moderate temp.

TABLE XVI.—*Stress-Deformation Tests.*

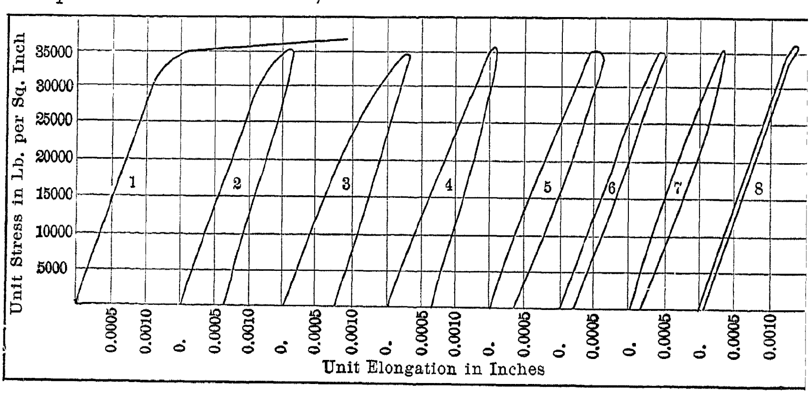
Specimen 11, round rod. Diam., 0.63 in. Area, 0.312 sq. in. Mild steel.

Specimen 12, ⅝ in. by 2 in. Area, 1.25 sq. in. Extra soft steel.

Load, Lb. Per Sq. In.	Specimen 11: Mean of Micrometer Readings (Inches).	Specimen 11: Elongation (Inches Per Inch)	Specimen 11: Remarks.	Specimen 12: Mean of Micrometer Readings (Inches).	Specimen 12: Elongation (Inches Per Inch).	Specimen 12: Remarks.
0	0.5964	0.00000		0.5225	0.00000	
3,200	0.5955	0.00011		0.5216	0.00011	
6,400	0 5947	0.00021		0.5207	0.00022	
9,600	0.5940	0.00030		0.5195	0.00037	
12,800	0.5932	0.00040		0.5184	0.00051	
16,000	0.5925	0.00049		0.5180	0.00056	
19,200	0.5911	0.00066		0.5170	0.00069	
22,400	0.5904	0.00075		0.5162	0.00079	
25,600	0.5892	0.00090		0.5151	0.00092	
27,200	No read	ing.		0.5148	[illegible]	
28,800	0.5884	[illegible]	Elastic limit, 30,000 lb. per sq. in.	0.5144	[illegible]	Elastic limit [illegible] lb. per sq. in.
30,400	0.5880	[illegible]		0.5139	0.00107	
32,000	0.5875	0.00111		0.5129	0 00120	
32,800	No read	ing.		0.5095	0.00162	
33,600	0.5865	0.00124		0.4675	0 00687	
35,200	0 5829	0.00167		No read	ing.	
36,800	0.5667	0.00371	Permanent stretch in 8 in. [illegible]	No read	ing.	Permanent stretch in 8 in. [illegible]
0	0 5758	[illegible]		[illegible] 4757	0 00585	

Specimen 11—*Mild Steel* (see Tables XVI. and XVII.).

Round rod. Diam., 0.63 in. Area, 0.312 sq. in. Elastic limit, 30,000 lb. per sq. in. Permanent set in 8 in., 0.02 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.

3.—73 hr. after overstrain, 72 hr. at freezing-temperature (1 hr. interval).

4.—2,708 hr. after overstrain, 2,707 hr. at freezing-temperature.

5.—2,732 hr. after overstrain, 2,707 hr. at freezing, 24 hr.
6.—2,754 hr. after overstrain, 2,707 hr. at freezing, 46 hr.
7.—2,778 hr. after overstrain. 2,707 hr. at freezing, 70 hr.
8.—2,801 hr. after overstrain, 2,707 hr. at freezing, 93 hr.

At moderate temperature after steam-bath.

TABLE XVII.—*Cyclical Loading of Specimen* 11—*Mild Steel.*

Round rod. Diam., 0.63 in. Area, 0.312 sq. in. Measurements on 8-in. length.

Load, Lb. Per Sq. In.	15 Minutes After Overstrain.		73 Hours After Overstrain, and 72 Hours at Freezing.		2708 Hours After Overstrain, and 2707 Hours at Freezing.		2732 Hours After Overstrain, 2708 at Freezing, 24 at Moderate After Steam Bath	
	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).
0	0.5767	0.00000	0.4904	0.00000	0.5851	0.00000	0.5771	0.00000
6,400	0.5748	0.00024	0.4885	0.00024	0.5828	0.00029	0.5748	0.00029
12,800	0.5729	0.00047	0.4864	0.00050	0.5811	0.00050	0.5735	0.00045
19,200	0.5714	0.00066	0.4842	0.00077	0.5790	0.00076	0.5716	0.00069
25,600	0.5696	0.00089	0.4816	0.00110	0.5768	0.00104	0.5692	0.00099
32,000	0.5675	0.00115	0.4785	0.00149	0.5746	0.00131	0.5652	0.00149
35,200	0.5632	0.00169	0.4763	0.00176	0.5725	0.00158	0.5647	0.00155
32,000	0.5645	0.00152	0.4768	0.00170	0.5730	0.00151	0.5647	0.00155
25,600	0.5657	0.00137	0.4785	0.00149	0.5746	0.00131	0.5664	0.00134
19,200	0.5675	0.00115	0.4802	0.00127	0.5756	0.00119	0.5682	0.00111
12,800	0.5693	0.00092	0.4820	0.00105	0.5772	0.00099	0.5706	0.00081
6,400	0.5709	0.00072	0.4835	0.00086	0.5787	0.00080	0.5723	0.00060
0	0.5728	0.00049	0.4856	0.00060	0.5795	0.00070	0.5747	0.00030

Load, Lb. Per Sq. In.	2754 Hours After Overstrain, 2708 at Freezing and 46 at Moderate After Steam Bath.		2778 Hours After Overstrain, 2708 at Freezing and 70 at Moderate After Steam Bath.		2801 Hours After Overstrain, 2708 at Freezing and 93 at Moderate After Steam Bath.	
0	0.5694	0.00000	0.5669	0.00000	0.5650	0.00000
6,400	0.5670	0.00030	0.5651	0.00022	0.5631	0.00024
12,800	0.5652	0.00052	0.5634	0.00044	0.5616	0.00042
19,200	0.5633	0.00076	0.5616	0.00066	0.5598	0.00065
25,600	0.5615	0.00099	0.5599	0.00087	0.5580	0.00087
32,000	0.5596	0.00122	0.5580	0.00111	0.5562	0.00110
35,200	0.5580	0.00142	0.5559	0.00137	0.5550	0.00125
32,000	0.5584	0.00137	0.5565	0.00130	0.5560	0.00112
25,600	0.5601	0.00116	0.5580	0.00111	0.5579	0.00089
19,200	0.5620	0.00092	0.5600	0.00086	0.5593	0.00071
12,800	0.5640	0.00067	0.5617	0.00065	0.5613	0.00046
6,400	0.5657	0.00046	0.5635	0.00042	0.5631	0.00024
0	0.5681	0.00016	0.5658	0.00014	0.5650	0.00000

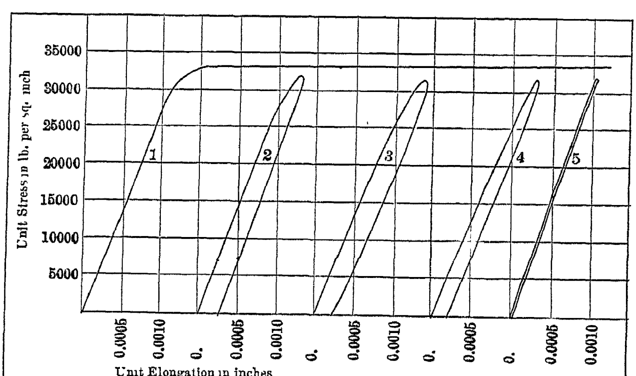
TABLE XVIII.—*Cyclical Loading of Specimen* 12—*Extra Soft Steel.*

Section, ⅝ in. by 2 in. Area, 1.25 sq. in. Measurements on 8-in. length.

	15 Minutes After Overstrain.		71 Hours After Overstrain, 70 Hours at Freezing.		2112 Hours After Overstrain, 2111 at Freezing.		2135 Hours After Overstrain, 2111 at Freezing, 24 at Moderate After Steam Bath.	
Load, Lb. Per Sq. In.	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch).	Mean of Micrometer Readings (Inches).	Elongation (Inches Per Inch)
0	0.5760	0.00000	0 4932	0.00000	0.4965	0.00000	0.5288	0.00000
3,200	0.5752	0.00010	0.4926	0.00007	0.4953	0.00015	0.5278	0.00012
6,400	0.5737	0.00029	0.4917	0.00019	0.4942	0.00029	0.5271	0.00021
9,600	0 5735	0.00031	0 4905	0.00034	0.4935	0.00038	0.5262	0.00032
12.800	0.5727	0.00041	0.4896	0.00045	0.4924	0.00051	0.5257	0.00039
16,000	0.5715	0.00056	0.4889	0.00054	0.4914	0 00064	0.5246	0.00052
19,200	0.5709	0.00064	0.4875	0.00071	0.4905	0 00075	0 5238	0.00062
22,400	0.5700	0.00075	0.4867	0.00081	0.4891	0.00092	0.5232	0 00070
25,600	0.5690	0.00087	0.4855	0.00096	0.4885	0.00100	0.5217	0.00089
28,800	0.5681	0.00099	0.4844	0.00110	0.4877	0.00110	0.5214	0.00092
32,000	0.5659	0.00126	0.4821	0.00140	0.4868	0.00121	0.5207	0.00101
28,800	0.5660	0.00125	0.4827	0.00131	0.4874	0.00114	0.5212	0 00095
25.600	0.5674	0 00107	0 4836	0 00120	0.4884	0.00101	0.5225	0 00079
22,400	0.5682	0.00097	0.4845	0.00109	0.4886	0.00099	0 5231	0.00071
19,200	0 5692	0.00085	0.4855	0.00096	0.4895	0.00087	0.5237	0 00064
16,000	0.5698	0.00077	0.4862	0.00087	0.4902	0.00079	0.5245	0.00054
12,800	0.5705	0.00069	0.4874	0.00072	0.4914	0.00064	0 5256	0.00040
9,600	0.5717	0.00054	0.4884	0 00060	0.4925	0.00050	0.5260	0.00035
6,400	0.5726	0.00042	0 4895	0.00046	0.4933	0.00040	0.5270	0 00022
3,200	0.5739	0.00026	0.4902	0.00037	0.4934	0.00039	0.5277	0 00014
0	0.5741	0.00024	0.4913	0.00024	0.4950	0 00019	0.5287	0.00001

Specimen 12—*Extra Soft Steel* (see Tables XVI. and XVIII.).

Section, ⅝ in. by 2 in. Area, 1.25 sq. in. Elastic limit, 28,500 lb. per sq. in. Permanent set in 8 in., 0.047 in.



1.—Stress-deformation curve.

Cyclical Loadings.

2.—15 min. after overstrain.

3.—71 hr. after overstrain, 70 hr. at freezing-temperature (1 hr. interval).

4.—2,112 hr. after overstrain, 2,111 hr. at freezing-temperature.

5.—2,135 hr. after overstrain, 2,111 hr. at freezing-temperature, and 23 hr. after steam-bath.

The Washoe Plant of the Anaconda Copper-Mining Co. in 1905.

BY L. S. AUSTIN, HOUGHTON, MICH.

(London Meeting, July, 1906.)

I. Introduction.

The Washoe plant,[1] in Anaconda, Mont., together with the local street-railroad, ranches, a foundry and machine-shop, a

[1] *Trans.*, xxxiv., 265.

brick-plant and the Montana Hotel, form a property under one management; to which should be added the mines in Butte owned by the company, from which part of the ore-supply of the plant comes. The Anaconda Copper-Mining Co., controlled by the Amalgamated Copper Co., was re-organized under the laws of Montana in 1893. A map of the plant, showing the numerous departments and illustrating the details of tracks, pipelines, and charging-stations for compressed-air locomotives, is given in Fig. 1.

II. Organization.

Apart from the general organization of the Anaconda Copper-Mining Co., already mentioned, the plant and accessories are under charge of the local manager, Mr. E. P. Mathewson. This is the largest non-ferrous metallurgical plant in the world, and illustrates, under the given conditions, the natural development resulting from the necessity of a complex organization.

With many smaller metallurgical plants the organization has generally been arranged so that the business manager gives his whole attention to business details, while the superintendent attends to the technical matters. With the Anaconda Copper-Mining Co., however, a technical man was chosen to attend to business details also; the wisdom of this arrangement has often shown itself, since, in this particular case, the mastery of business detail has been as satisfactory as when performed by one who had always confined his attention to business only. Moreover, a comprehensive view of the whole process by a master of technical detail has proved equally gratifying. For example, the 50-ft. reverberatory furnaces of three years ago were successively increased in length to 65, 85 and 102 ft., since it was not known what length of furnace could be used. To make these changes required the expenditure of $120,000 (all of which was recovered out of the economies attained during the course of the experimenting), and success required a complete knowledge of the technical side of the problem, as well as a clear idea of what the finances of the company would permit. Again, in increasing the length of the blast-furnaces at one time from 15 to 51 ft., it was necessary to know whether it was possible to remove a jacket while the blast was on, and to cut out or to repair one part of the furnace while the rest was in operation. At the time the change was made copper was

selling at a profitable figure, and it would not do to shut down in order to make changes. It was therefore necessary to make the change with practically no delay in running. With such considerations on the financial side, the manager knew he must advance only along lines involving no delay; had he to do with the technical side alone, he might have put up with the apparent necessity of shutting down so as to make the improvement.

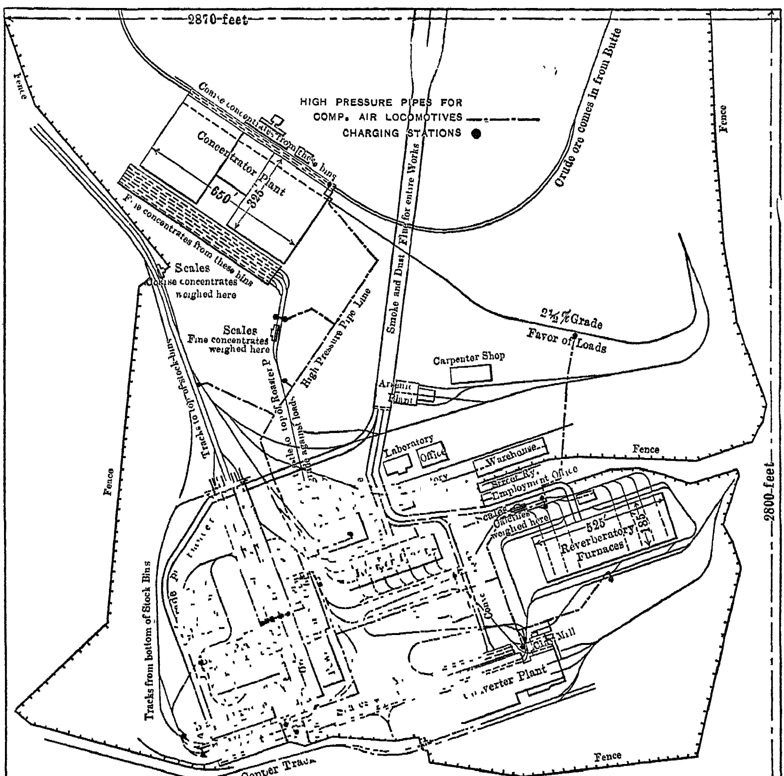


FIG. 1.—PLAN OF THE WASHOE PLANT OF THE ANACONDA COPPER-MINING CO., ANACONDA, MONT.

In order that the manager might give the closest attention to technical questions, an assistant superintendent was appointed to advise with him and with the heads of departments, and to take his place during his necessary absences.

The different departments about the works are under charge of foremen or superintendents, who are specialists in their lines of work, and who work in harmony with one another. There

are superintendents for the concentrating-mill, sampling-departments, power-houses, blast-furnaces, converters and the arsenic-plant; also chief foremen for the reverberatories, the roasters, the briquetting-plant and the slime-ponds. In addition, there is a general foreman of the works, who has full charge and authority in the absence of the superintendents of the departments. The 8-hr. system, in use at Anaconda, requires that there shall be three shifts daily, so that the general foreman is relieved by two foremen who assume his duties on the other shifts and who relieve each other on the afternoon and night shifts every two weeks, while the general foreman always keeps the morning or day shift. The departments having superintendents also have foremen, a head-foreman holding the morning shift, and two reliefs alternating on the other two shifts. The chief, or morning foreman, has the larger measure of authority and of responsibility. Thus, at no time is any department left uncontrolled, since the foremen relieve one another, and, as an additional precaution, the general foreman attends not only to the general supervision of the plant but also to those matters belonging to no particular department. Repairs and general construction are under supervision of the mechanical superintendent with an assistant and a head-mason. The complex handling of materials is in charge of a master of transportation. For experimental and control work there is an engineer of tests. A chief chemist reports all assays and determinations. The accounting is supervised by a chief clerk, and the purchasing of supplies, both for mine and works, is in charge of a purchasing-agent.

III. Production.

The various improvements in the Washoe plant have resulted in a largely increased production in 1905 above that of the preceding year. In 1904 there was treated daily 5,500 tons of smelting- and concentrating-ore, yielding an output of 350,000 lb. copper. In 1905 the plant handled 7,000 tons of ore daily, the resultant output of copper being 500,000 lb. The following was the average monthly output in 1905: Copper, 15,000,000 lb., @ 18.53c. per lb., $2,780,000; gold, $95,000; silver, $432,000; total, $3,307,000.[2]

[2] *Anaconda Standard*, December 25, 1905 (corrected).

The monthly pay-roll during 1905 amounted to $215,000. The average number of men employed in all departments at Anaconda was 2,450. The following data of daily operations may be of interest: Ore treated, 7,000 tons; coal consumed, 600 tons; coke consumed, 400 tons; lime-rock used, 1,600 tons; flue-dust produced, 190 tons; slag and tailings produced, 9,000 tons; yield of copper, 500,000 lb.

IV. Transportation.[3]

About 11,700 tons of ores, lime-rock, coke and coal are brought in daily by railroad-cars, the switching being done on tracks of the Butte, Anaconda & Pacific railroad. The ores and lime-rock come in 50-ton hopper-bottom cars, the coke in box-cars, and the coal in hopper-bottom or gondola cars. Deliveries of ores and limestone are made to bins or pockets, having outlet-chutes for drawing off the ore. Counting rehandled materials twice, more than 13,000 tons are handled about the works by means of 13 compressed-air locomotives, 12 weighing 13 tons each, and one, 21 tons. Each locomotive carries two storage-tanks for its air-supply, the air being taken from a pressure-system of pipes laid conveniently to the tracks, and having stations at which the locomotive stops to get its air-supply. This supply is carried at from 800 to 900 lb. per sq. in. A reducing-valve between the storage-tanks and the cylinder reduces the pressure to 150 lb. A fresh supply of air is taken at times ranging from 20 to 60 minutes. For this particular service, where the distances run are short, and where the cars are frequently stopped and started, the compressed-air locomotives have been most satisfactory, for convenience, reliability and simplicity of operation. They have been in constant service since 1900, and have needed only the natural running-repairs. They are operated over tracks aggregating 48 miles in length, which are distributed over the side-hill location of the works, about half a mile square. Were electric locomotives used, the overhead trolley-system would be a most complicated one. Moreover, the present locomotives have been able to stand the hardest and roughest usage with but little injury. Under like conditions, one could be assured that electric locomotives would have had

[3] *Cassier's Magazine*, vol. xxviii., pp. 466 to 478 (1905).

coils burned out, short-circuiting, and other troubles, apart from the danger arising from the wires themselves. The work must be done at all hours of the day and night, under the most varied weather conditions, and often surrounded by fumes of sulphur and dust. Endless spotting and shifting has to be done in order to weigh, load and unload the material handled. For example, in the case of the locomotive hauling the matte-ladles, they must be spotted for weighing three times on each round trip, in addition to three other stops, two for loading and unloading, and one to set the ladle on a side-track for cleaning. In the case of the 18-charge car-trains, each car of 2.5 tons capacity, the five ingredients of the charge must be weighed separately, and each two cars must be spotted to receive the ore from the chutes, and weighed for each ingredient. Each train must be moved and set eight or ten times while unloading. If steam-locomotives had been used, the smoke and steam in the buildings and under the bins would have interfered seriously with the signaling to the engine-runner.

The pipe-system, which serves to supply air to the various stations for the use of the locomotives, is composed of pipes varying from 6 in. to 2 in. in size. Even at the high pressure of from 800 to 900 lb. per sq. in., it has developed practically no leaks. The air is furnished by two 4-stage air-compressors, having cross-compound steam-cylinders equipped with Corliss valve-gear. The compressors are equipped with automatic regulators to insure constant high-pressure air.

V. Sampling.

The sampling-mill, furnished with two Brunton automatic sampling-machines, is used principally for sampling the concentrating and first-class or smelting-ore from the mines of the company and from others affiliated with the Amalgamated Copper Co.

Of the concentrating-ore, every fifth car is reserved for regular sampling, while all the smelting-ore is sampled, and the finer portion, under $\frac{3}{8}$ in. in size, is screened out as first-class fines, to be made, together with other ingredients, into briquettes.

The mill adjoins a high-level track, by which ore is brought to discharge into hopper-bottomed steel-lined bins, from which

the mill supply is taken. Fig. 2 shows, diagrammatically, the course of the ore through the mill, and from it, it will be noticed that four cuts are taken out by four automatic sampling-machines, the ore being crushed finer after each cut, and before the new cut is taken out. In the figure, the rejected portion of the ore is represented as being delivered to a car upon the

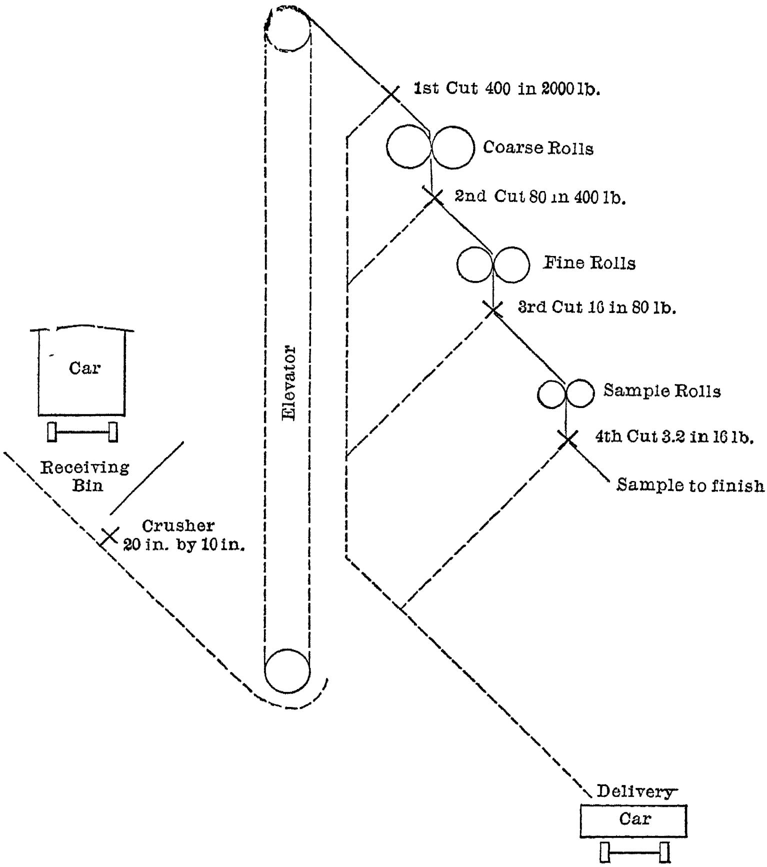


FIG. 2.—FLOW-SHEET AT AUTOMATIC SAMPLING-MILL.

low-level track adjoining the mill. As a matter of fact, this portion is raised by elevator to trommels, in order to remove the fines. In consequence, a ton of 2,000 lb. is represented by 3.2 lb., or one 625th of the original portion taken. Thus, in a 500-ton lot there would remain 1,600 lb. of a sample. The portion so obtained is mixed in a pile and quartered down by means of a Brunton quartering-shovel. The reduced sam-

ple is ground through a sampling-grinder of the Leadville-Engelbach type, after which it is mixed, and further reduced by riffles. The grinding-plate, Fig. 3, upon which the pulp is ground to pass an 80- to 100-mesh screen, has three raised sides and rounded corners. The muller used in the sampling-room is shown in Fig. 4.

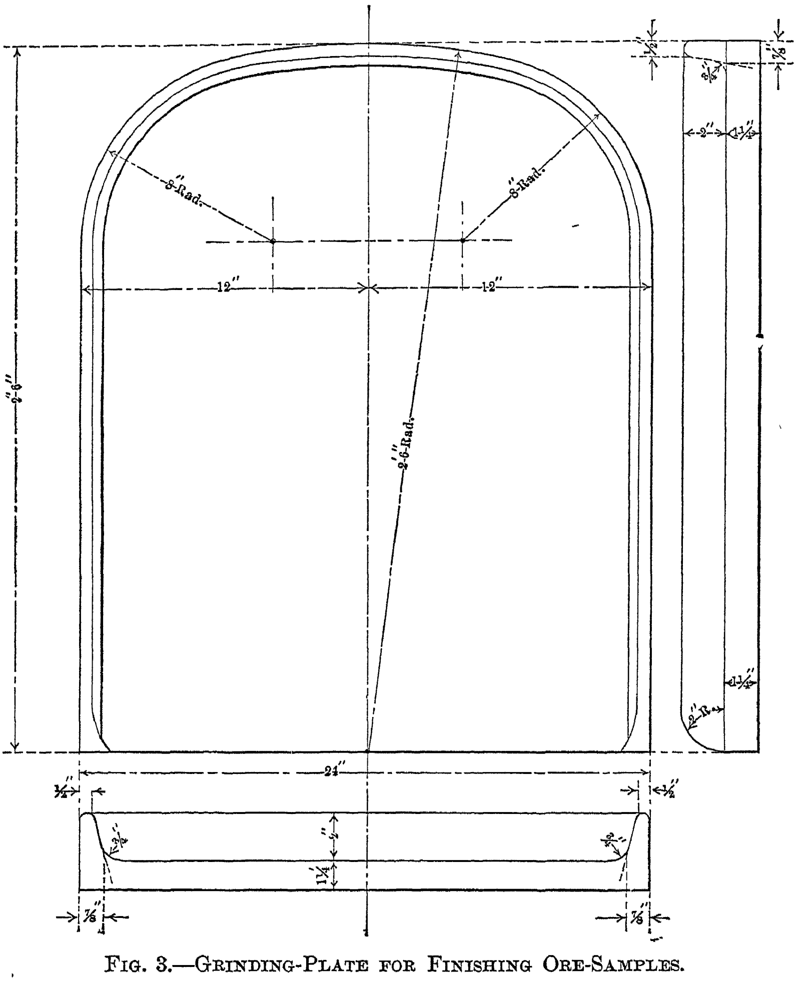


FIG. 3.—GRINDING-PLATE FOR FINISHING ORE-SAMPLES.

Samples of all metallurgical products about the works are handled at the laboratory sampling-plant—a sampling-mill situated centrally of the works. About 9,000 samples monthly are taken at different stages of operations, and include ore, concentrates, tailings, calcines, slag, matte, flue-dust, converter-

copper and anodes. A control of all the metallurgical work is thus obtained, and, when combined with the systematic weighing of all charges, insures a complete control of the operations

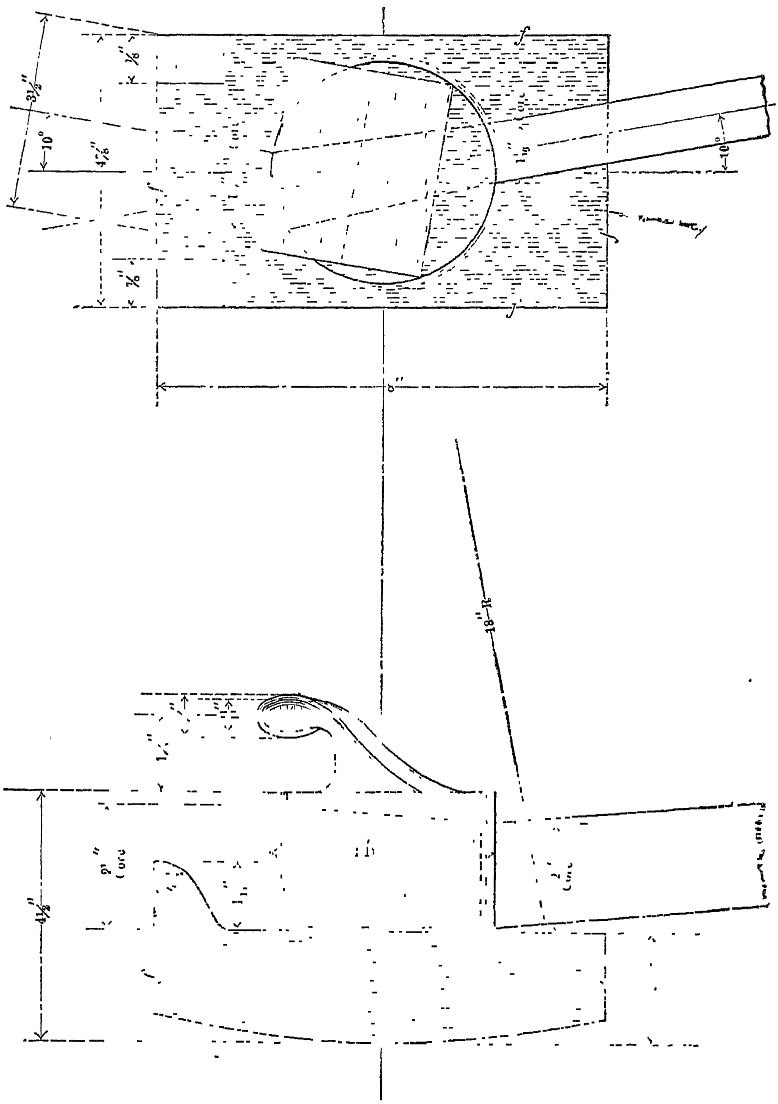


FIG. 4.—MULLER FOR GRINDING ORE-SAMPLES.

and the detection and knowledge of all losses. Many of the samples taken are small "grab" samples, as well as those taken systematically. Consequently, the machinery is simple, the grinding being done in sample-grinders and finished on buck-

ing-plates. The samples are dried out, either upon a steam-bath, 4 ft. by 12 ft. in area, or in a steam-closet, where they are placed in pans between racks of steam-coils. As fast as they are finished, the rejected portions of samples are sent to an elevated bin for eventual return to the works.

VI. Concentration.

The concentrating-plant, elsewhere described by Mr. M. Schwerin,[4] has a side-hill or terrace location, and adjoins the high-level track and storage-bins, from which it draws its supply of ore. It consists of two mills, with a power-house between. Each mill has four sections, so there are eight in all. The capacity of a section may be given at 1,000 tons per day of 24 hr., and it may be estimated that 7.5 sections are constantly running, or that one section is down half of the time for repairs or adjustments. This would give a daily capacity, when the ore supply is sufficient, of 7,500 tons. A maximum output of 8,250 tons, however, has been attained. There is a large power- and boiler-plant between the two halves of the concentrator building, but much of this installation will be thrown out of use when the company obtains its power by electric installation from the plant of Montana Water E. P. Co., 22 miles distant, and the Missouri River Power Co., near Helena, 75 miles away.

To supply the eight units, already specified, there are eight storage-bins, each of 1,200 tons capacity. These bins have flat bottoms, the ore forming its own slope in them, so that the wear of the bin is slight.

The following outline describes the flow of the ore through one section of the mill:

(1) Hopper-bottomed cars of 50 tons capacity to (2).

(2) Storage- or supply-bin of 1,200 tons to (3).

(3) Shaking feed-trough, having a screen-bottom of 1.25-in. round holes, oversize to (4), undersize to (7).

(4) Coarse crusher, 12 in. by 24 in., to (5).

(5) Two trommels, each 36 in. in diameter by 5 ft. long, and having 1.25-in. round holes, oversize to (6), undersize to (7).

(6) Two crushers, each 5 by 15 in., to (7).

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[4] *Engineering and Mining Journal*, vol. lxxvi., p. 388 (Sept. 12, 1903).

(7) Two elevators, having 15-in. belt, buckets 7 by 14 in., speed per min. 450 in., to (8).

(8) Two trommels, 36 in. in diameter by 6 ft. long, and having $\frac{7}{8}$-in. round holes, oversize to (9), undersize to (13).

(9) Two 2-compartment Harz jigs on 1.25-in. feed, concentrates to (12), middlings to (16), through (15), (8) and (13), hutch product to (14).

(10) Four Harz jigs.

(11) One coarse roll, 42 by 15 in., 1,147 ft. peripheral speed, to (13), through (8).

(12) Coarse-concentrates bin, concentrates to blast-furnace in hopper-bottom cars.

(13) Two 7-mm. trommels, 36 in. in diameter by 8 ft. long, oversize to (10), undersize to (16).

(14) Twelve 2-compartment Evans jigs on 7-mm. feed, size of screen 24 by 41 in., concentrates to (32), middlings to (35), hutch product to (32).

(15). One fine roll, 42 by 15 in., 1,147 ft. peripheral speed, to (16), through (8) and (13).

(16) Four 5-mm. trommels, 36 in. in diameter by 6 ft. long, making 20 rev. per min., oversize to (14), undersize to (18).

(17) Twelve 2-compartment Evans jigs, 5-mm. feed, concentrates to (32), middlings to (35).

(18) Four 2.5-mm. trommels, 36 in. in diameter by 7.5 ft. long, oversize to (17), undersize to (19).

(19) Twelve 2.5-mm. Evans jigs, concentrates to (32), middlings to (35).

(20) One middlings-roll, 42 by 15 in., to (21).

(21) Four 1.5- by 12-mm. diagonal-slot trommels, 36 in. in diameter by 6 ft. long, oversize to (20), undersize to (22).

(22) Four 3-spigot separators, spigot-product to (23), overflow to (26).

(23) Eighteen 3-compartment middling-jigs, 1.5-mm. feed, concentrates to (32), middlings to (24), tailings to dump.

(24) Three 6-ft. Huntington mills, 53 rev. per min., 62 tons capacity, grinding to pass a 1- by 12-mm. slot-screen, to (25).

(25) Eighteen 3-compartment Huntington mill finishing-jigs, 1-mm. feed, concentrates to (32), middlings to (24), tailings to dump.

(26) Sixteen settling-tanks, 5 ft. wide by 4 ft. deep by 18 ft. long, spigot-product to (27), overflow to (28).

(27) Eighteen Wilfley tables, concentrates to (32), tailings to dump, slime to (29), middlings to (34).

(28) Slime-ponds situated below the reduction-works.

(29) One large tank, spigot-product to one Wilfley table, (30), overflow to (28).

(30) One Wilfley table, concentrates to (32), tailings to dump.

(31) Elevator to (32)

(32) Settling-tanks, concentrates to roasting-furnaces, overflow to (28).

(33) Eighteen Wilfley tables, concentrates to (32), tailings to dump, slime to (28).

(34) Eight tanks, spigot-product to (33), slime to (28).

It will be noticed, in this scheme of separation, that no product of the coarse jigs is allowed to go to waste, but that all the portion going over the tail-boards of these jigs is re-crushed and sized for further jigging. No tailings go to waste larger than 1.5 mm. Screening by trommels is carried as far as practicable, after which the spigot-products of classifiers are used on the fine jigs. Settling-tanks are used for unwatering, and for classifying the unwatered product. All fine concentrates from the fine jigs and from the Wilfley tables pass to an extended series of boxes in which the concentrates settle out, while the supernatant water slowly passes through the boxes, dropping most of its load, the settled product being drawn off at the bottom of the boxes through water-tight gates or valves. The final muddy water, however, is carried half a mile by launders to the slime-ponds, elsewhere described. The coarse concentrates go to a set of hopper-bottom bins, from which they are drawn off to cars, and carried to the storage-bins at the blast-furnaces.

The concentrating-ore is practically chalcocite, enargite and pyrite in a quartz and aluminous gangue.

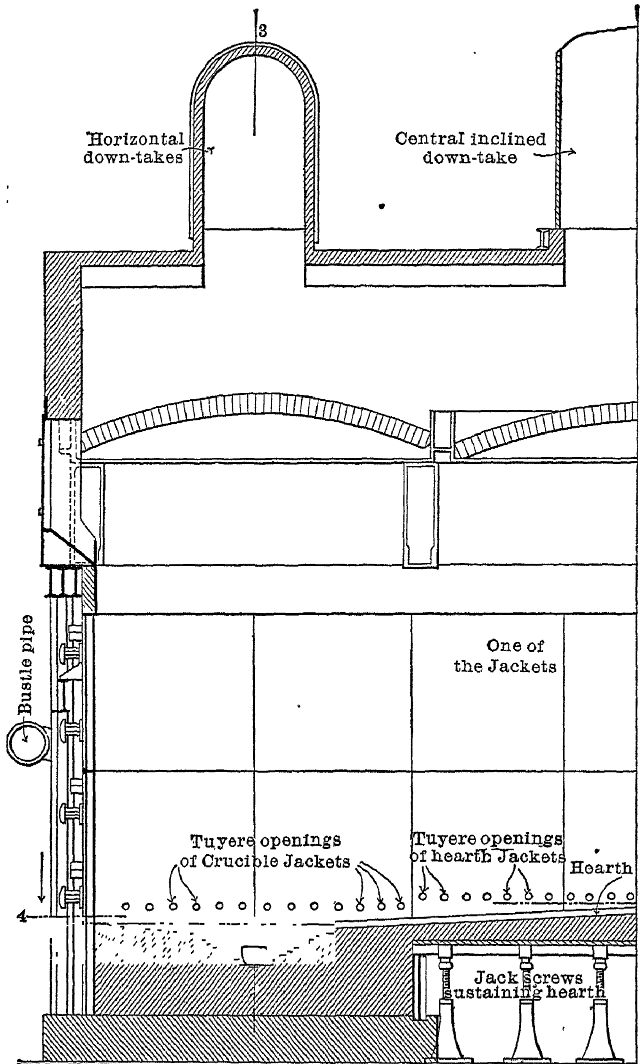
VII. Blast-Furnace Plant.

1. *Construction.*—The blast-furnace building formerly contained seven furnaces, each 56- by 180-in. area at the tuyeres. These were arranged with their longer axes parallel to the length of the building, for convenience of charging by means of track charge-cars. A new furnace was made by joining

furnaces Nos. 1 and 2 and including the 21-ft. space between them, which gave a furnace 51 ft. long, and of the original width of 56 in., as shown in Fig. 5. In a similar manner furnaces Nos. 3 and 4, and Nos. 5 and 6 were united; at present there are three large-sized furnaces, each 51 ft. long, and one furnace, 180 in., or 15 ft. long. At the slag-floor level are seven fore-hearths, or settlers, as arranged for the original furnaces, which serve equally well for the enlarged furnaces, there being two fore-hearths for each. A matte-track, at a lower level, serves to bring in the matte-ladles, into which the matte of the fore-hearths is tapped. Fig. 6 illustrates a cross-section of the enlarged furnace, and Fig. 7 a cross-section of the building. Views of the interior of the blast-furnace from the levels of the feed-floor and the hearth are given in Figs. 8 and 9. The large blast-furnaces, 56- by 612-in. tuyere-dimensions, have side-boshes of 8 in., making the width at the throat 6 ft. There is no end-bosh. There are two tiers of jackets, each 7 ft. 5. in. high. In either row, and on either side, the three central jackets are 7 ft. wide each, and the four remaining ones are 7 ft. 6 in. each, making thus the total length of 51 ft. The concrete foundation is surmounted by the two original crucibles, to which was added the connecting bridge, the latter formed of water-cooled plates supported by short jack-screws, as shown in Fig. 5. The crucibles, or sumps, as they may be aptly called, slope to the tap-hole. The bridge portion has a hearth sloping either way from the center to the crucibles. Thus the total drainage gravitates to either tap-hole. It may happen that the hearth becomes obstructed so as to throw more of the flow to one of the crucibles and less to the other. Such a result, however, need not interfere with the effective operation of the fore-hearths. Both crucible and hearth are made of silica brick, which has been found to be very enduring with the 40-per cent. silica slag here produced.

Above the upper tier of jackets comes the heavy mantel-plate, 2 ft. 3 in. high, having a sloping front, and surmounted by the apron or receiving-plates, 21 in. high, with a slope of 45°, both together making a hopper upon which the charge slides down, tending to keep the finer portion closer to the walls, and the coarser material at the center of the furnace, all of which helps to promote regular working.

The furnace has a closed top, the fumes being carried off by three down-takes to the large dust-chamber adjoining the blast-furnace building. There are 46 tuyeres at the back, and 42 at

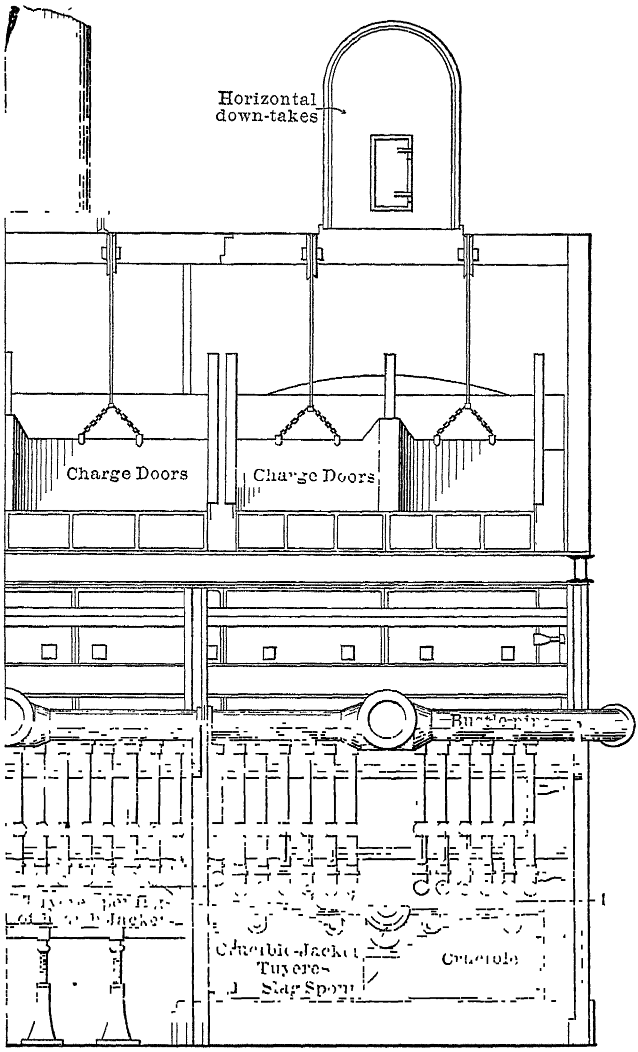


Half Longitudinal Sectional Elevation

FIG. 5.—SECTION AND ELEVATION OF

the front side of the furnace, or 88 in all, set at 14-in. centers. The space supplementing the lesser number of tuyeres at the front is utilized to make room for the two slag-spouts. It is customary to plug, or leave off, the four tuyeres nearest the

ends of the furnace, as this is found somewhat to prevent accretions, so that, practically, only 84 tuyeres are used. The omission of the two tuyeres over the spouts does not appear to



Half Longitudinal Elevation.

THE 56- BY 612-IN. BLAST-FURNACE.

interfere with the proper operation of the furnace at this point, all of which indicates irregular working of tuyeres. Where it has been undertaken to cut out every alternate tuyere the furnace has not worked so well. This, indeed, may be due to the fact

that some of the tuyeres cut out were in good working position, or, rather, that the more frequent tuyeres mean more

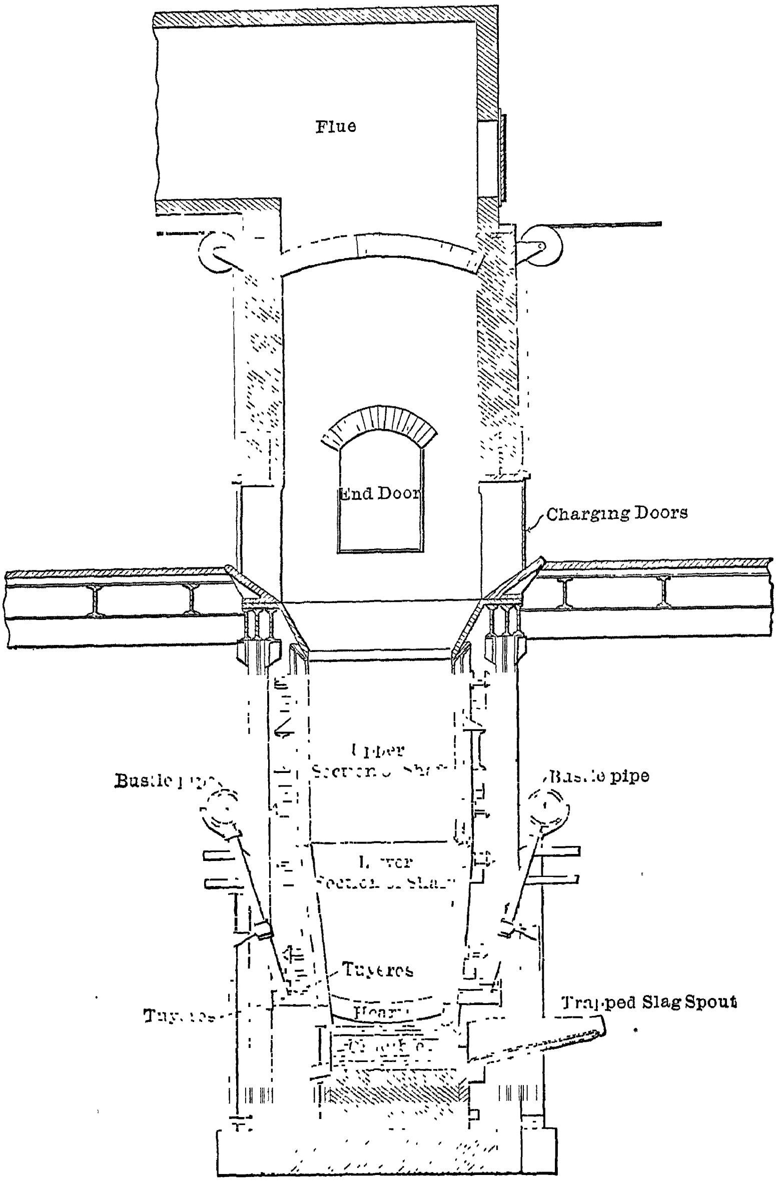


FIG. 6.—TRANSVERSE SECTIONAL ELEVATION OF 56- BY 612-IN. BLAST-FURNACE.

chances of getting air into the furnace. The tuyeres are punched several times on a shift, and it can hardly be sup-

posed that all the air enters the furnace by the small 1-in. hole thus formed. No doubt a good deal gets in between the jacket

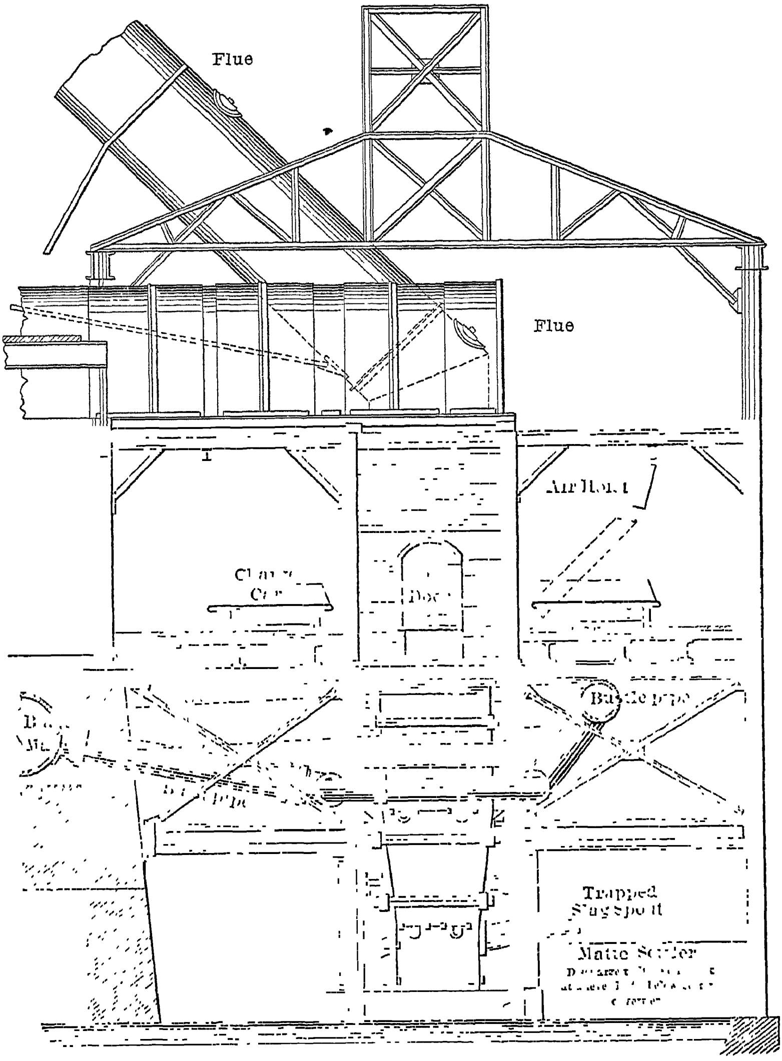


FIG. 7.—CROSS-SECTION OF FURNACE BUILDING AND ELEVATION OF 56- BY 612-IN. BLAST-FURNACE.

and the crust, eventually finding its way by devious passages (those of the least resistance) to the interior of the furnace.

The bustle-pipe extends completely around the furnace, with two branches from the blast-main, each with its own shut-off gate, so that, to shut down the furnace, two valves must be

closed. The tuyere-pipes are 5 in. in diameter; the tuyeres being bolted to the jacket for easy removal in case of necessity. The connection to the blast-pipe is made by a galvanized iron sleeve, with a ring-clamp to make the joint tight. The fore-hearths, or settlers, of the former smaller furnaces, serve equally well for the large ones. They are 16 ft. in diameter and 5 ft. high, the matte-tap being 3 ft. below the top of the fore-hearth. The slag enters on the side adjoining the furnace, and flows out on the opposite or front side. Experiments at Great Falls show the best point of outflow to be opposite the point of entrance. The discharging slag flows down a steeply-inclined spout, which delivers it opposite a 6-in., flattened water-jet situated just below the floor. This is supplied by jacket and other water. Besides this, there is a flow of water from the encircling cooling-pipe of the fore-hearth, which contributes to the stream that sweeps away the granulated slag through a semicircular cast-iron lined launder, and conveys it to the dumping-point on the flats below the works.

The exterior of the furnace, having so long a side-wall, is braced and prevented from bulging by ties, carried around the furnace both at the level of the top of the lower jacket, and also under the floor. The horizontal pull of this bracing at the corner posts of the furnace is resisted by inclined ties attached to the floor above at adjoining corner posts, as shown on Fig. 7. It will be noticed that, between the three panels forming the length of the furnace, the jackets are supported by I-beams of the length of the panel (22 ft.). Two sets of the I-beams are to resist outward pressure of the jackets, the upper and heavier set to hold up the upper jackets when the lower ones have, for any reason, been removed. Two of the outlet, or down-take, flues (soon to be changed to resemble the central one) are of steel, lined with brick, and have hopper-bottoms for the removal of the flue-dust. The third, or central down-take, has an upward and then a downward slope, having a pitch so steep that no flue-dust can rest upon it. Flue-dust, on the upward-sloping flue, slides back to the furnace; on the downward-sloping side it goes to the dust-chamber.

The dust-chamber, 40 ft. wide and 280 ft. long, is provided with sheet-steel pyramidal hoppers, by which the flue-dust is conveniently drawn off to the charge-cars when they are run in

below. This chamber, 20 ft. high, affords a liberal space for the settling of the flue-dust. It is connected to the stack by

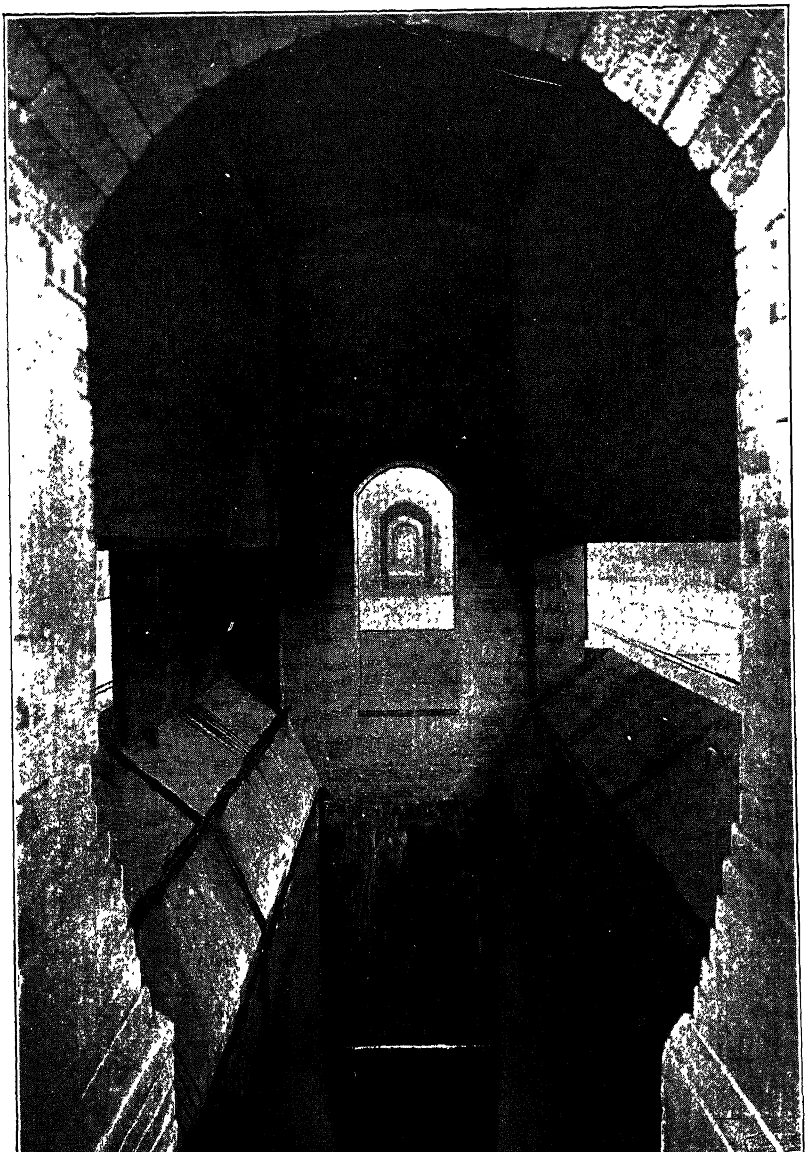


FIG. 8.—INTERIOR OF BLAST-FURNACE AT FEED-FLOOR LEVEL.

means of a flue of 20 ft. by 15 ft., or 300 sq. ft. area of cross-section.

2. *Operation.*—The furnace is charged through seven doors on each of the long sides, giving access to the entire length.

FIG. 9.—INTERIOR OF BLAST-FURNACE AT HEARTH-LEVEL.

It also has end doors for barring-out. The charge is brought in by a train of 18 cars at a time, each car holding 2.5 tons. Eight of these cars constitute a charge, amounting to from 18

to 20 tons. The coke is charged separately in 2-wheeled buggies, each of 30 cu. ft. capacity, and capable of holding 1000 lb. The amount used is 10 per cent. of the charge.

The weighing of charges to an 18-car train is thus performed: The cars are spotted to the coarse-concentrates bins, where the specified quantity is weighed in (to two at a time), the flow being controlled by a slide-gate operated by hand. A swinging chute, operated by a compressed-air cylinder, also assists in controlling the discharge. After the concentrates, the 18 cars are simultaneously charged with the briquettes, as more fully mentioned in the description of the briquetting-plant. The train is then taken to the first-class ore-bins for the next item of the charge, then to the slag-bins, and the limestone-bins. It will be seen that, in this way of putting in the materials, the finer portion is at the bottom of the car, and it is so charged into the furnace that it falls to the near side, while the coarser material goes to the center portion of the furnace.

The temperature of the gases from all the furnaces, in the flue just beyond the dust-chamber, drops as much as 62° C., and the draft-pressure drops 0.2 in. of water when six of the 46 doors are open. The air, with a draft-pressure of 1.7 in. in the flue, has a velocity of 1.5 ft. per sec. through the doors (each of 20 sq. ft. area), and, with six doors open, an admission of 10,800 cu. ft. per min., or one-fifth of the total out-going gases. The quantity entering is proportional directly to the number of doors opened. Of the total heat developed by the burning of the coke and sulphur, 18.6 per cent. escapes in the outgoing gases, which, at the entrance to the down-take, have a temperature of 380° C. Not more than these six doors need be opened at any one time, either for charging or for cutting accretions or crusts from the furnace.

With regard to the influence of limestone, comprising approximately one-third of the charge, whose decomposition into CaO and CO_2, completed at 800° C., abstracts heat at the surface of the charge, and thus lessens overfire: such decomposition takes 425 pound-calories per pound of limestone, or, as 3.5 lb. limestone is used per pound of coke consumed, it absorbs 1,487 calories out of the total of 6,680 calories developed by the combustion of this pound of coke and the 1 lb. of sulphur, which is disposed of at the same time.

On the other hand, at the zone of fusion, the CaO, in forming a silicate, evolves 297 calories, where such an evolution is most to be desired, resulting in an outflowing slag which reaches a temperature of 1,150° C.

The temperature of the escaping gases varies from 100° C., in a freshly-charged furnace, to 560° just before a new charge is put in, especially when there has been a longer interval than usual before again charging. These variations become greater as the jackets become more crusted over, such accretions forming to the thickness of several inches.

The addition of a fresh charge results in a drop of temperature of 200° C., while the opening of the doors on one side cools the gases 80° C., the difference, 120°, being due to the putting in of the cold charge. There results a gradual rise of temperature of 25° C. per min., the rise being retarded because of the endothermic action of the limestone, the evaporation of the water, and the heat absorbed in heating up the charge.

It would be well to note here, that, with a fusible self-fluxing charge containing but little limestone, and with so high a percentage of coke, this rise of temperature would be more rapid, and overfire would increase to such a degree, indeed, as to promote the greater formation of accretions which tend to crust over the top of the charge and to delay, if not stop, smelting.

With regard to the charging-operation, I am not at liberty to give the proportions or analyses of the constituents of the charge at the Washoe plant, but, in Table I., I present a typical Montana blast-furnace charge made up from data in Prof. Hofman's paper, Notes on the Metallurgy of Copper of Montana.[5] The analyses of matte and slag are from the same source, and from Peters's book on Copper-Smelting (7th ed., p. 552). Compared with the charge calculation given by him on p. 367, there is but little flue-dust made. To-day, all fine concentrates are sent to the roasters and then to the reverberatory furnaces. The fine ore is also screened out from the first-class ore and is worked into briquettes. Thus the flue-dust loss has been cut from 18.75 per cent., which he mentions, down to 3 per cent. in present practice. The grade of the matte has also been re-

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[5] *Trans.*, xxxiv., 258 to 316 (1904).

TABLE I.—*Charge-Sheet.*

Name of Ore.	Moisture.	Weight.		Cu.		SiO_2.		Fe and Mn.		CaO and MgO.		S.	
		Wet.	Dry.	Per Ct	Wt.	Per Ct.	Wt.	Per Ct.	Wt.	Per Ct.	Wt.	Per Ct.	Wt.
Coarse concentrates.....	3.0	1,550	1,500	8.7	130	22.0	330	27.0	405	.....	....	35.7	535
Briquettes....................	8.0	1,620	1,500	6.9	104	45.0	675	11.0	165	[illegible]	[illegible]	17.0	255
First-class ore...............	3.0	2,060	2,000	8.0	160	50.0	1,000	12.1	[illegible]	[illegible]	[illegible]	[illegible]	308
Converter-slag.............	......	1,500	1,500	2.0	30	30.0	450	42.9	[illegible]	[illegible]	[illegible]	[illegible]	10
Limestone......	......	3,500	3,500	.. ..	.....	5.0	175	1.0	35	51.0	1,785	......	....
Coke.........................	......	....	1,020	....	......	6.9	70	1.2	12	0.8	8	0.7	7
		10,230			424		2,700		1,502		1,818		1,115

Weight of slag, $\frac{2,700}{40 \text{ per cent.}} = 6,750$ lb., or 66 per cent. of charge.

Cu in slag, 6,750 by 0.33 per cent. = 22 lb., or Cu_2O = 25 lb.

Cu in matte, 424 — 22 = 402 lb., and the matte weighs $\frac{402}{40 \text{ per cent.}} = 1,005$.

Fe in matte, 1,005 by 30 per cent. = 301 lb., leaving for the slag 1,502 — 301 = 1,201 lb., or of FeO 1,544 lb.

S in matte, 1,005 by 23 per cent. = 231 lb., and in slag 26 lb., or in both 257 lb.

S volatilized 1,115 — 257 = 859 lb., or 77 per cent. of the total sulphur.

From these data we have in the slag SiO_2 2,700 lb. (40 per cent.) ; FeO, 1,526 lb. (22.5 per cent.) ; CaO, 1,818 lb. (26.9 per cent.) ; Cu_2O, 25 lb. (0.4 per cent.) ; S, 26 lb. (0.4 per cent.) ; other bases, by difference, 9.8 per cent.

From Peters[6] we have taken a matte, and used it in our calculation, of the composition, S, 23 per cent. (231 lb.) ; Cu, 40 per cent. (402 lb.) ; Fe, 30 per cent. (301 lb.) ; other bases, 7 per cent. (71 lb.).

The matte-fall, based upon the entire charge of 10,230 lb., is 10 per cent., and the concentration, 5 tons of ore into one of 40-per cent. matte.

duced from 50 per cent., the practice at the time the book was written, to 40 per cent., as now made. It is now possible to work 40-per cent. copper-matte successfully in the converter, because of the firmer quality of the lining due to the power-rammers used. The loss by volatilization, 80 per cent. of sulphur, has, in spite of the large amount present, resulted in giving a 40-per cent. matte. This has been brought about, in part, by the use of a high-pressure blast of 40 oz. per sq. in., which shows itself in much overfire and by the violent agitation of the materials at the surface of the charge.

The high pressure of 40 oz. is necessary because of the small openings by which the air enters the furnace, consisting of holes no more than 1 in. in diameter, where the slag has been pierced by the punching-bar, and of accidental seams between the jacket and its adherent crust, as well as by cracks and

[6] *Modern American Methods of Copper Smelting*, by E. D. Peters, Jr., 7th ed. p. 552 (1895).

seams in this crust. The estimate allows for no leakages about the tuyeres, and the sound of the escaping air indicates there is some loss from this source, a defect due to faults of construction, since, with the much higher pressures carried by the iron blast-furnace, the furnace-room is comparatively quiet. It may also be concluded that, with a more unrestrained entrance for the air, the pressure might well drop to half its present value, and that, moreover, with twice the volume of air now entering the furnace, twice the tonnage could be put through. It is true, in the case of these large furnaces, that improvement has been incompletely met in other details. The same feeble methods are used to-day in punching the tuyeres, and in barring-down the large furnaces, that have been used for many years. Improvement in the delivery of air at the tuyeres is the direction in which improvement will best be repaid. It would result in a furnace needing a low smelting-column, in the dropping of pressure and, with it, the saving of power, as well as the ability of the furnace to increase its tonnage with the increase of the air admitted.

Crusts or accretions are to-day removed by attack from the feed-floor level by a gang of three or four men, who, exposed to the high radiant heat and fumes at that point, feebly work a heavy bar to break down these accretions. I believe, however, that metallurgists are well aware of these defects, and at this particular plant they will endeavor to remove them, as soon as they can get to it. It must be remembered that the first of those 51-ft. furnaces has been in operation only since March, 1904, less than a year before the present writing. There seems now, however, but little chance of the solution of the problem of improved air-supply into the furnace when one takes into consideration its difficulties.

As indicated in Fig. 7, there is a track at a level 10 ft. below the slag-floor. The matte-ladles are run in and spotted to the desired furnace, and the fore-hearth tapped to the ladle. A dolly, having a 6-in. button-head, carries the conical clay stopper, made of a local clay. A full ladle of matte of 10 tons is drawn off. The flow is stopped with the dolly, the tapper pressing it in with all his force, while an assistant strikes the head of the dolly. The opening having been stopped, the tapping-bar is gently driven through the fresh clay, remaining

there until the next tapping. The launder, leading the matte to the ladle, is 12 in. wide, with a semicircular bottom lined with clay. There is an independent launder to the other settler also, so that it is possible to tap from both fore-hearths to the same ladle. The ladle, when spotted, is ventilated by a hood suspended above it, which, terminating in a straight stack through the roof above, carries off the fumes.

It has sometimes, though seldom, happened, as after a shut-down, that the contents of the fore-hearth have become cooled. Directly after tapping the matte, and before the settler has again filled with slag, it has been warmed by blowing compressed-air into the settler, so as to bessemerize the molten matte still remaining in the settler. The reaction, being exothermic, serves to raise the temperature of the molten contents.

VIII. Advantages of Large Furnaces.

Numerous advantages, in the operation of the larger furnaces over the ordinary ones, show themselves upon study.

1. *Saving in Fuel.*—The large furnaces will work with 10 per cent. of coke where 11 per cent. has been needed for the 15-ft. ones. This arises from the fact that, while the sectional area at the tuyeres has increased 3.4 times, the radiating-surface at the sides up to the top-surface of the charge (10 ft.) has increased but 2.8 times. Taking into account the lessened percentage of fuel, still 6 per cent. more is burned in unit time, thus increasing the maximum temperature and giving more satisfactory fusion. Mr. Mathewson, the local manager, thinks that in lead-smelting it should be possible to save 2 per cent. out of the 14 per cent. generally used.

2. *Saving in Jacket-Water.*—While the jacketed surface has been increased 2.8 times, the capacity has increased 3.4 times.

3. *Quick and Large Discharge of Matte and Slag.*—It will be noticed that the crucible is contracted in volume as compared with other practice on smaller furnaces. As a result, the contents of the crucible are more quickly discharged. The increased tonnage smelted gives an increased flow, so that there is less chance of obstructions forming, or, if formed, they are swept away in the large volume of slag and low-grade corrosive matte. The increased intensity of combustion, already alluded to, results in a hotter slag (1,170° C.); so markedly true is this,

that the 40-per cent. silica slag, usually carried, appears perfectly fluid, flowing with as much ease as I have seen with the 35-per cent. silica slags of ordinary practice. The high silica shows itself, however, upon withdrawing from the flow with an iron rod a portion which strings out in a characteristic manner. Indeed, it seems to me that the assigned limit of 40 per cent. might be far exceeded, were it not probable that tonnage would be reduced, because of the slower driving of the more siliceous slags. Whether, in order to save limestone, or rather to increase capacity by its omission, slags of 60 per cent. or more will ever be found profitable, as in Mansfeld practice, is an interesting question. The temperature of the slag leaving has dropped, on an average, 50° C. The temperature of the matte remains much the same as when it enters. Thus we have temperature of entering slag and matte, 1,170°; of the slag leaving the settler, 1,120°; and the outflowing matte, when tapped to the ladles, 1,170°, and sometimes a little higher, indicating possible exothermic reaction going on within the forehearth.

4. *Decrease of Incrustation.*—Comparing the two sizes of furnace, the larger one may be considered as being made by the union of three, placed end to end, a construction which was carried out by keeping the two adjoining furnaces running, while the intermediate portion was being built. At the last moment, and during a short shut-down, the dividing-walls were removed and the whole thrown into one. Thus we may reckon that four end-walls, out of the six needed in the three smaller furnaces, have been eliminated. This has by so much lessened the surfaces for incrustation, especially since it is at the ends of the furnace that the most trouble of this kind occurs. In fact, on the long side-walls accretions have little support, and it is necessary only to let the charge down, when the side-accretions will fall forward into the furnace. Bridging, however, may sometimes take place, thus holding-up the side-crusts. In such a case the bridge is smelted away, care being taken that when it goes, so much does not fall forward as to obstruct the furnace-operations at that point. With the heavy blast-pressure carried (40 oz.), the surface of the charge is in intense agitation, accompanied, just before charging, with much overfire and the projection of pieces of coke and materials of the charge. Hence

the importance of screening, in the case of all the first-class ore used, which, at the time of sampling, is passed over a trommel having $\frac{3}{8}$-in. holes, the undersize going to the briquetting-plant. About 3 per cent. of flue-dust is caught in the dust-chambers, while a good deal of the projected material falls back into the furnace. The closed top promotes this action, since it gives a place for dust to settle. The charge-doors, when opened, also have a cooling and settling effect. The flue-dust caught in the chamber is sent to the reverberatories..

5. *Elasticity of Operation.*—As a preliminary to the operation of a large furnace, it was necessary to find out if it could be kept running while part of it was undergoing cleaning-out or repair. To this end a jacket was removed while the blast-pressure was kept on and the furnace was continued in operation. The crust, coating the jacket at that point, held back the heat, so that while the jacket was 7.5 ft. wide by 7 ft. high, it could yet be removed. This operation was much assisted by means of a compressed-air crab, both in pulling out the old and in hoisting the new jacket into place. On account of the demand for continuous operation, it was necessary to wait two or three months before a jacket had to be removed because of a leak. It was also found that, by removing adjoining tuyeres and the proper jacket, the dead portion of a hearth obstruction could be got at without serious trouble to the men. Where the heat showed too fiercely, a sheet of iron supported by bars served to protect the spot. The dead material having been removed from the crucible, the jacket was again put up, and this portion of the furnace again put into operation. This, fortunately, has not frequently been necessary. Of course, all this diminishes the effective smelting-area, but it would cut proportionally less figure in a long than in a short furnace. It may be said, therefore, on account of the possibility of making repairs, and of cutting-out while the furnace is still smelting, that campaigns may be considered endless, being terminated only if the supply of ore becomes exhausted.

6. *Large Flow Through Fore-Hearths.*—The capacity of the large furnace is 3.4 times that of the smaller one; hence, nearly twice as much of the mixed matte and slag enters the fore-hearth in a given time. Despite the increased flow, an im-

proved settling and a cleaner slag have resulted, due to its greater fluidity. The increased flow has also tended to keep the fore-hearth hotter. No more labor is needed because of the doubled flow. Certainly, the matte has to be tapped more frequently, but the same men can accomplish that also, since they have only 15 tappings to make in 24 hours. The slag, being granulated, takes no more labor, though more water is needed for granulating it. A fore-hearth or slag-spout is changed by stopping the flow at that tap-hole during the time. The united flow, therefore, goes through the other fore-hearth, which is able to perform the duty of both.

7. *Alteration Without Interruption.*—At the Washoe plant, as already mentioned, the longitudinal axes of the furnaces are parallel to the length of the building, a disposition suited to charging by charge-cars in trains. In consequence, it was possible to put in the bridge, jackets, top and down-take of the 21-ft. connecting portion. The end-jackets of the original furnaces were then removed, the crusts broken away, and the entire furnace brought into operation with practically no delay, at a time when output was all-important.

8. *Variations in the Composition of Slag.*—The part of the hearth most exposed to corrosion is the central, bridged portion. To lessen this action, the charge for that part of the furnace may be made more siliceous than normal, the ends less so; thus the average silica of the slag would be unaffected. Again, were it desired to put in a basic charge at one end of the furnace in order to smelt out a crust, this could be done without in any way affecting the other end of the furnace. In fact, it would be possible to run three different slags at the same time.

9. *Less Labor-Cost.*—As just shown, the cost for tappers and furnace-tenders at the slag-floor level is less; and at the charge-floor, where power-feeding is used, the resultant cost for feeders is less also. On the other hand, the labor per ton for bringing in coke is not lessened. The present way of handling the coke is the cheapest and most practical, however, since the coke is taken direct from the cars in which it comes to the works, and is handled with less breakage than by any other system. In the practice of the Washoe plant the surface of the charge is

carried 6 ft. below the charging-door, giving a 10-ft. smelting-column, so that there is, undoubtedly, some breakage in dumping the coke direct from the coke-buggies into the furnace. It may be said, however, in defense of the practice, that it is followed because it is found that, when the charge is shot in from the charge-car from that height, some segregation occurs, resulting in the finer ore going to the side and the coarser to the middle line of the furnace, as it should do. The labor for the original seven 15-ft. furnaces may be given as follows per shift of 8 hours:

Upper floor:			
7 feeders,	@	$4.00	$28.00
8 coke-wheelers,	@	3.00	24.00
2 charge-loaders,	@	3.00	6.00
4 bin-men,	@	3.00	12.00
1 extra man,	@	3.00	3.00
Lower floor:			
1 foreman,	@	5.00	5.00
1 head-tapper,	@	4.00	4.00
7 tappers,	@	4.00	28.00
10 tapper-helpers,	@	3.00	30.00
1 flume watchman,	@	3.00	3.00
			$143.00

Or, for three shifts, $429 for an output of 2,940 tons. To this may be added 11 per cent. of coke @ $8.50 per ton, or $2,748.90, making a total for fuel and labor of $1.08 per ton. On the other hand, operating the three large (51-ft.) and one small (15-ft.) furnace, for labor per shift is obtained:

Upper floor:			
4 feeders,	@	$4.00	$16.00
4 feeder-helpers,	@	3.00	12.00
13 coke-wheelers,	@	3.00	39.00
2 charge-loaders,	@	3.00	6.00
4 bin-men,	@	3.00	12.00
1 extra man,	@	3.00	3.00
Lower floor:			
1 foreman,	@	5.00	5.00
1 head-tapper,	@	4.00	4.00
7 tappers,	@	4.00	28.00
10 tapper-helpers,	@	3.00	30.00
1 flume watchman,	@	3.00	3.00
			$158.00

Or, for three shifts, $474 for an output of 5,220 tons. The coke

consumed, 10 per cent., at $8.50 per ton, costs $4,437, making the total for fuel and labor $0.94 per ton. This is a saving of 13 per cent. upon those two items, being $730.80 per day.

10. *Less Initial Cost.*—The cost of a furnace with 3.4 times the capacity would not exceed from 2.5 to 3 times that of the smaller one. There are two fore-hearths and three down-takes for the 51-ft. furnace. The bracing of the long sides, in this particular instance, has been accompanied with but little additional expense, taking advantage of the trussing and the construction already existing. The same building serves to shelter the larger furnaces as effectually as it did the smaller ones; that is to say, the same structure is enough for an additional output of 2,280 tons, or 77 per cent. more than the original capacity.

IX. Briquetting-Plant.

The building formerly contained two Chisholm-Boyd and two White briquetting-presses of a capacity of 125 tons each in 24 hr. To operate one machine and to handle and dry the briquettes, there were needed 8 men per shift, including the foreman, or 24 men daily. The briquettes were dried in an adjoining shed.

At present, the company has four end-cut, auger brick-machines, one being constantly in use, and having, for the single machine, a capacity of 840 tons of briquettes per 24 hours.

The briquettes are composed of one-third ore (screened from the first-class ore), one-third slimes from the slime-ponds of the works, the balance of so-called table-concentrates (the product of settling-tanks at the concentrator), and 5 per cent. of washed reverberatory coke. The washed coke is made from the ash-pit droppings of the reverberatories. It is conveyed from the ash-pit, in a stream of water, to the coke-washing plant, where the ashes and clinkers are jigged out, and a tailings of coke, of about chestnut size, is recovered. This material, worked into the brick, tends to give them an open texture, favoring the escape of the moisture quietly rather than explosively, as would be the case were the structure dense. The coke thus incorporated can be figured upon as replacing 40 tons daily of the more expensive oven-coke at the blast-furnace.

These materials are drawn from inclined-bottom storage-bins to a conveyor-belt passing in front of the four hopper-chutes.

A man stands at each of three of the chutes to regulate their discharge, while the discharge of the coke-bin is regulated from a distance by the pug-mill man, who is standing at the pug-mill. The three men at the chutes become quite expert at sending along the proportion assigned to them, and their work is controlled by the pug-mill man, who also signals when they are to stop feeding.

The conveyor-belt discharges to another at right angles to it, which conveys the mixture to the supply-chute of the pug-mill, which is a double-spindle, horizontal mill making 22 rev. per min., in which the various ingredients are mixed with just enough water for molding in a brick-press situated below. The latter is a No. 7 end-cut, worm-feed brick-machine, made by Chambers Bros. Co., Philadelphia, Pa. When first put in operation, from 400 to 500 tons were produced in 24 hr., but, with greater skill and better regulation, the output has been raised to a daily average of 840 tons, or to a maximum of 880 tons. To operate the plant there are needed 9 men per shift, including the foreman. The bricks weigh from 5 to 10 lb. each, and the machine, in full operation, can produce 160 bricks per minute. The bricks are again conveyed by a 10-in. flat rubber belt, to another, parallel to the long side of the building and commanding a set of hoppers placed directly over the center of a track, which is inside the building and is heated by steam. There are 36 of these hoppers, each capable of holding a ton of briquettes. Deflecting-boards sweep the briquettes into the hoppers, the man in attendance operating the deflectors so as to fill first the even-numbered, and then the odd-numbered hoppers. The hoppers are each 2 ft. 6 in. wide at the top, 2 ft. wide at the bottom, 3 ft. 4 in. long and 2 ft. 6 in. deep. They are steel-lined and have drop bottoms. The bottoms can be dropped simultaneously, or individually, as shown in Fig. 10. In this illustration the contents of two hoppers only have been dropped. In winter the room is heated by steam so that the briquettes will not freeze. Each alternate hopper is painted red, the others white; corresponding red and white lights indicate which set is full. When the hoppers are empty, no light shows.

A train of from 16 to 18 charge-cars is loaded as follows: About 200 lb. of coarse concentrates is drawn off to the cars from the

coarse-concentrate bins. The train is then spotted under the hoppers as indicated by the lights to take the charge of briquettes, and the bottoms of the proper bins are dropped together, thus filling the cars with one movement of a lever. The coarse concentrates are put into the cars first, in order to prevent the briquettes, which are fresh from the press, from sticking to the bottom. When it is desired to omit the briquettes from any desired car, that particular lever can be thrown out of gear. The hoppers are so spaced that the alternate ones come over the centers of the charge-cars. The work goes on with great precision, and 18 cars, if desired, can be filled with a single movement of the lever.

It is to be noticed that these briquettes are not handled at all. They are used moist, not being dried. They crumble somewhat, but still stand handling well enough for smelting.

X. Roasting-Plant.

1. *Kilns.*—The roaster building contains 56 McDougall roasters, as improved by Evans and Klepetko. There are 14 rows, each of four furnaces, each group of four being driven from a common shaft, and any furnace operated independently by its own friction-clutch. There are four groups of furnaces, each driven by a motor, which receives its current from the central power-house.

Professor Hofman[7] gives a careful description of the roasting-furnaces of this plant. To this I add certain details of practice.[8] At the beginning of each 8-hr. shift the workman stops the furnace, and removes the lumps of ore or crusts which have accumulated, especially on the rabbles and roof of the third or "cutting" floor and on the roof of the fourth floor, to which they adhere. The lumps thus removed furnish material well-suited to blast-furnace smelting. Then, starting the furnace, the few remaining lumps are taken out as they are swept round by the rabbles. Finally the doors, except the bottom ones, are closed, and the furnace resumes its roasting. This work takes about an hour. The side drop-holes of the even-numbered hearths, each situated in front of a door, are cleaned by means of a 1-in. chisel-pointed bar 4 ft. long, and having its cutting-

[7] Notes on the Metallurgy of Copper of Montana, *Trans.*, xxxiv., 277 (1904).

[8] See also *Mineral Industry*, vol. xii., p. 98 (1903).

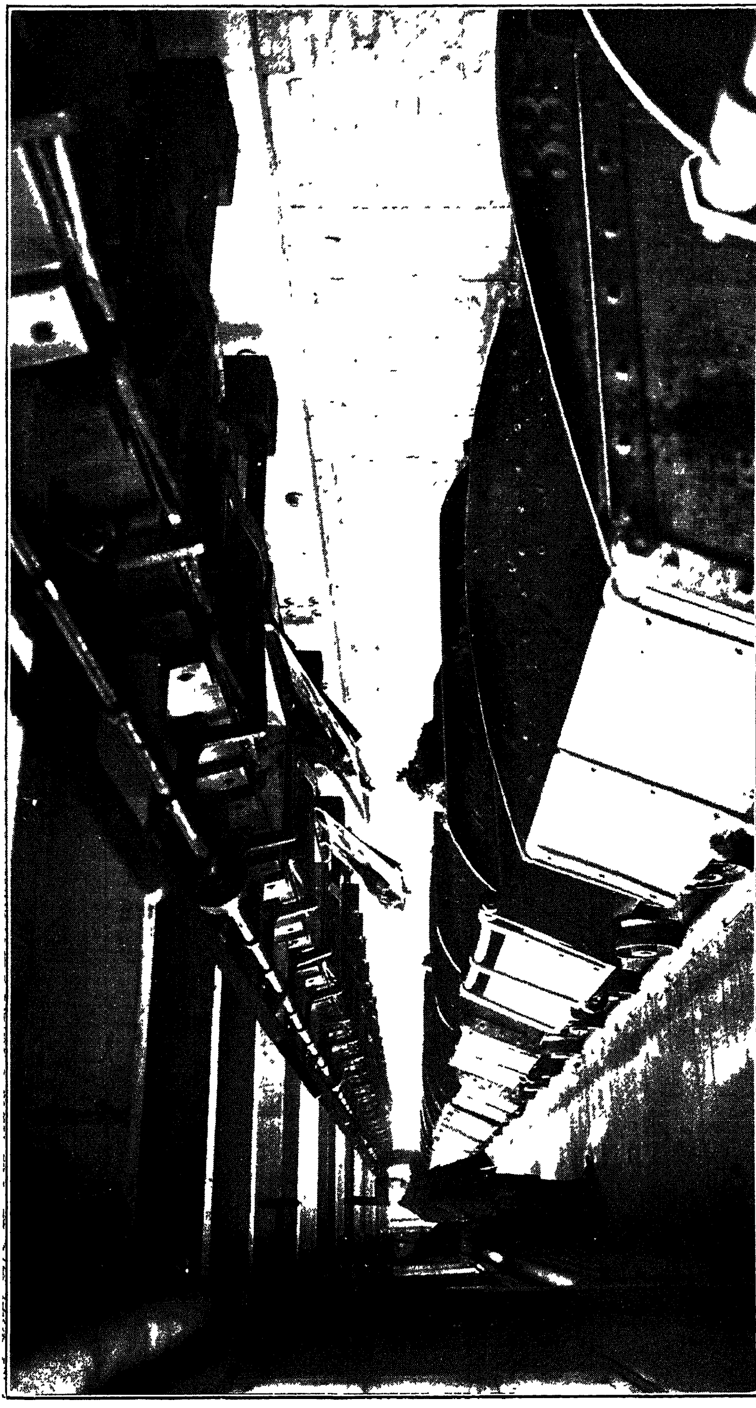

FIG. 10.—BRIQUETTE-LOADING DEVICE, WITH TRAIN IN POSITION FOR LOADING.

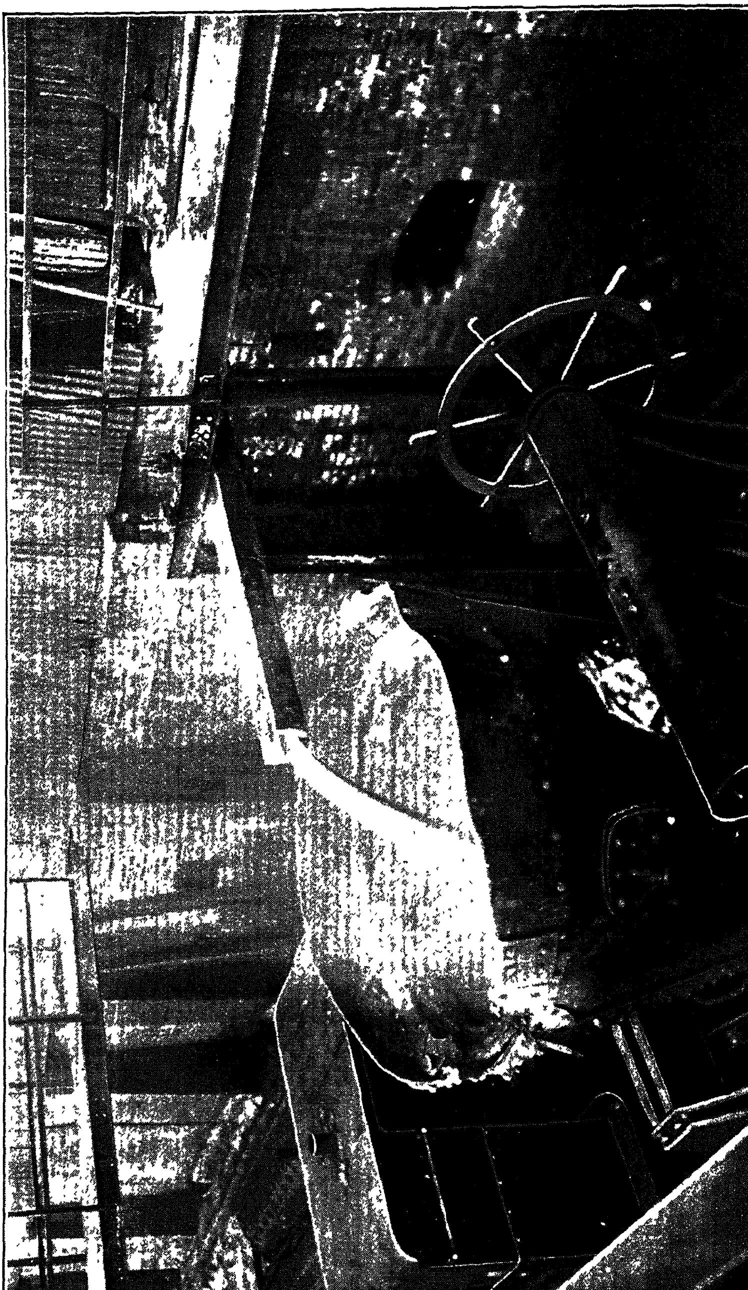

FIG. 11.—TAPPING MATTE FROM REVERBERATORY FURNACES INTO 10-TON LADLE.

end curved so as better to reach the sides of the drop-hole. The ore collecting on the central shaft, and at the central draft-holes of the odd-numbered hearths, is quite easily removed because of the water-cooled surfaces to which it adheres. In the older installation it was customary to draw the flue-dust from the cross-flue to the calcine-hopper of the furnace. Whenever calcines were drawn off, this flue-dust would escape from the hoppers and act with corrosive effect upon the skin of the workman. The cross-flue was accordingly made over with a hopper-bottom, by which the flue-dust, as it collected, was quietly withdrawn to calcine-cars, and sent back to be again fed through the furnace. This did away with a serious difficulty in operating.

The cross-flues are made of 0.25-in. plate-steel, but they have been so corroded that they are being replaced by flues of brick, having also hopper-bottoms for the convenient removal of the flue-dust. The rabble-blades are 8 in. long and wear to 6 in. before being replaced. When new, they clear the hearth by 2 in. The central drop-hole allows 16 in. between its edge and the shaft. The ore, plowed by the rabble above, drops upon the shield of the next lower rabble-arms at a point midway between them. In so doing, it showers down through the ascending air, which actively roasts it, but, at the same time, this air-current carries away the finer particles of the ore as flue-dust. Hence the considerable portion of that material made, which varies from 3 to 5 per cent.

When a furnace is to be shut down for repair, the feed is stopped and the hearths are emptied by the continued operation of the rabbles. Upon starting up again, about two firings of wood are put on the bottom and fifth hearths. This is followed by coal. When the wood is gone the rabbles are started, as well as the feeding of fresh ore until it comes down to the fourth or fifth hearth. At the same time coal is added to the hearths as they heat up, until the furnace has become sufficiently hot to start the burning of the ore. The heat creeps up and the furnace gradually gets into operation.

The sulphur left in the roasted ore, based upon a capacity of 40 tons daily, is from 6.5 to 8.5 per cent., but preferably 7 per cent. The charge consists of 27.5 tons of fine concentrates, 1.25 tons of finely-crushed limestone, 1.25 tons of returned flue-

dust. The actual output, however, is as high as 47 tons daily. The concentrates may be replaced, in part, by fine screenings from first-class and fine table-concentrates. Fig. 12 shows the progress of the roasting on the successive hearths. It shows that about the same duty is performed by each hearth after the first.

The gases escaping from the upper hearth have a tempera-

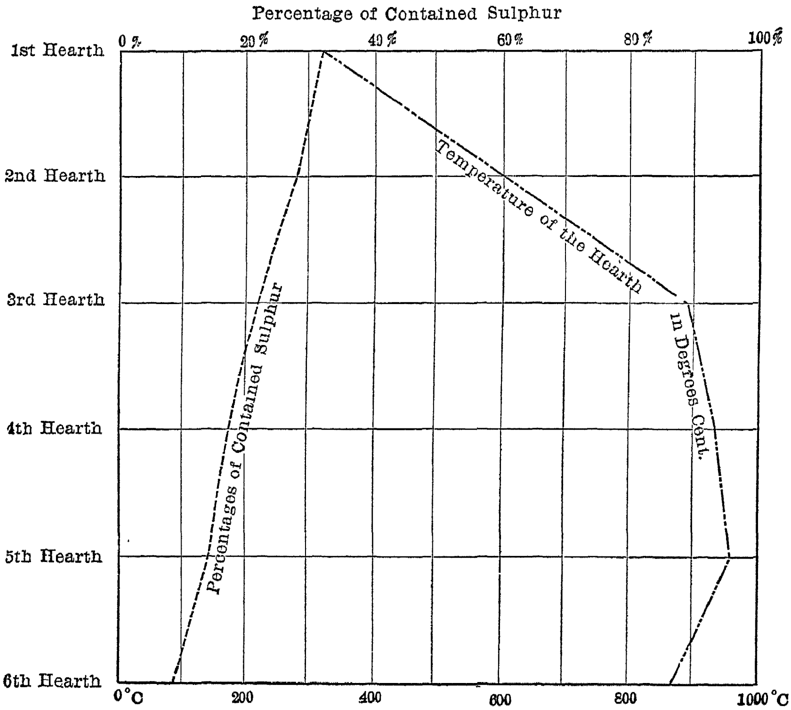

FIG. 12.—PROGRESS OF REACTIONS AND FLAME-TEMPERATURE OF THE McDOUGALL ROASTER.

ture of 315° C. and contain 2.25 per cent. of SO_2 by volume. By the time they reach the flue-chamber they have cooled to 117° C. The appearance of the roasting at the different hearths is as follows:[9]

On the first hearth, the ore, dropped at the circumference, and containing 6 per cent. of moisture, is drying out, but attains to no visible heat. Entering the second hearth it still looks dark, but shows a blue flame by the time it reaches the

[9] *Trans.*, xxxiv., 277 (1904).

borders of the hearth, where it is 600° C. On the third hearth, some sparks show where the rabble passes, together with blue flame becoming hotter, with more abundant sparking, and with a flame-temperature of 900° On the fourth hearth, the sparking has ceased, the ore having attained an orange heat. In falling upon this hearth from the one above, the ore, as it showers down, burns freely, and, undoubtedly, the roasting is hastened, even by this momentary, but thorough, exposure to the ascending air. On the fifth hearth, the sulphur is leaving the ore, the discharge temperature being less than the entering one; that is, it is brighter near the outer periphery. Here the maximum temperature of 960° C. is attained. On the sixth, and final, hearth, the heat has become uniform but lower (860° C.). As the ore leaves the hearth it seems brighter, but speedily cools off to 660° C. as it falls, smoking freely, into the hopper. The calcines are withdrawn rather promptly from the hoppers, so that, by the time they enter the reverberatory furnace, they will average 420° C. The Evans-Klepetko type of roaster, here used, has water-cooled rabble-arms, but it has been found that these abstract but little heat, and in no way hinder the roast. Ore, sticking to the water-cooled surfaces, adheres but lightly, and is easily removed. The percentage of sulphur given in the roast-and-temperature diagram, Fig. 12, is 32, but, in general, the average is 34 per cent.

In considering the rate of roasting, the amount of sulphur removed is about the same on each hearth. However, the velocity of expulsion of sulphur should decrease in the ratio of the quantity remaining, much as interest payments decrease on deferred payments. Interfering with this principle is the fact that much heat is used in the expulsion of moisture on the upper hearth, and that the ore, which needs to be heated to a certain temperature (300° C.) before it gets started to roasting, there fails to get such a start. On the lower hearth, roasting still proceeds actively, because, there, the ore gets air which is quite fresh, while the escaping gases contain SO_2 to the extent of 2.25 per cent. of their volume, or 5 per cent. of their weight. No doubt, tonnage could be increased by blowing in hot, fresh air at the upper hearth, but this would be adding to the expense, so that it is just as well that it is principally a drying-hearth. The ore takes more than 2 hr. 15 min. to pass

through the furnace. To the ore has been added 4 per cent. of fine limestone, the calcines also containing 2.5 per cent. of CaO.

The composition of the escaping gases is:

	By Weight. Per Cent.	By Volume. Per Cent.
SO_2,	4.95	2.25
SO_3,	1.46	0.53
O,	19.60	18.45
N,	74.00	78.77

Thirty-two pounds of air is needed per pound of sulphur, and, since 13.3 lb. of the latter is burned off per minute, there is needed in that time 6,384 cu. ft., which passes up the central hearth-opening at the rate of 8.8 ft. per sec.; hence, the quantity of flue-dust produced amounts to, at least, 5 per cent. A screen-analysis shows the fineness of the product—viz., on 10-mesh screen, 9.7; between 10- and 30-mesh, 25.3; between 30- and 80-mesh, 30.7; passing 80-mesh, 33.4; total, 99.1.

2. *Reverberatory Furnaces.*—There are seven reverberatories, the largest of the kind in the world, placed in two buildings. Each furnace supplies waste heat enough for two 300-h.p. Stirling water-tube boilers. The gases, after passing through the boilers, are conveyed by the reverberatory flue to the main flue. Whenever it is desired to clean the boilers, a by-pass flue conveys the escaping gases direct to the reverberatory flue, the connection through the boiler being meanwhile shut off.

A sectional plan and elevation of this style of furnace is given in Fig. 13, and the details of construction are as follows: Length of hearth, No. 1, 115.8 ft.; of Nos. 2, 4 and 5, 102.5 ft.; and of Nos. 3, 6 and 7, 112.5 ft.; all hearths are 19 ft. wide, and have fire-boxes 16 ft. by 7 ft., or 112 sq. ft. area. Referring to Furnace No. 1, Fig. 13, the outlet-flue at the neck is 60 by 38 in., or of 16 sq. ft. area. The skimming-door, 12 in. by 15 in. wide, has its skim-plate 4 in. above the hearth at that point. The hearth, however, has a slope of 8 in. toward the matte tap-holes. At 24 ft. from its front end, the furnace begins to draw into the jamb, where it is 7 ft. wide and 4 ft. high to the crown of the arch. The rise of the arch is 1 in. to the foot, or at the middle section 19 in. In the roof there were 10 transverse expansion-joints, 3 in. wide, or 30 in. in all, left in the furnace when built. These, however, are now each a little more than 1 in. when the furnace is cold. In the bridge

itself there are three transverse expansion-joints, each of 4 or 5 in., which close up when the furnace is fully heated. The bridge is 4 ft. across, and is 27 in. above the hearth and 24 in. above the grate. It has a hollow conker-plate through which air passes by a ventilating-passage to the draft-flue, thus keeping the bridge cool. At this end of the furnace the height from the hearth to the crown of the arch is 6.7 ft. Twenty checker-holes, each 3 by 3 in., provided for the admission of air, are located transversely to the furnace over the fire-bridge.

In some of the furnaces additional checker-holes have been made at the front of the fire-box for additional air, but for other furnaces this extra supply is not needed. In order to

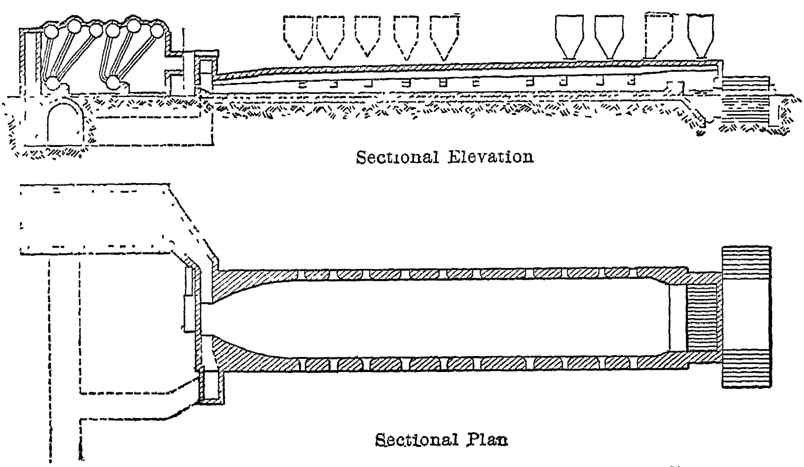

FIG. 13.—REVERBERATORY FURNACE NO. 1, HAVING 115.8 FT. LENGTH OF HEARTH, WITH WASTE-HEAT BOILER-SETTING.

determine the height of the coal in the fire-box, two circular stoke-holes, 12 in. in diameter, are placed in the front wall of the fire-box, and these opening are closed by wheel-doors. Over the middle line of the fire-box are four charge-openings, 12 in. square, through which the four coal-hoppers discharge into the fire-box. There are ten side-doors, each 8 by 15 in., with sills 18 in. above the level of the skim-plate at the front. The sides of the furnace and the fire-bridge are stepped off for several courses on the inside, thus strengthening the lower portion of the walls and the bridge. There are four charge-openings in the roof, having circular covers. Through these openings the charge from the two hoppers above drops into the furnace.

The operation of these furnaces is as follows: In the fire-box of a furnace, which has been shut down for repairs, a fire of wood is put, followed by an increasing firing of coal, until, in 48 hr., the first charge may be dropped in. As fast as charges are melted, others are added, until the furnace has been filled. The furnace takes from 10 to 15 days to get up to its full heat. The charges of 15 tons are put in, on the average, every 80 min., 10 tons being dropped from the first hopper and 5 tons at the second one, 90 ft. from the front of the furnace. It must be noted that the charge drops upon a bath of a minimum depth of from 3 to 4 in. of matte with 8 in. of slag above it, so that there is little chance of the new charge sticking to the hearth. The hot calcines, of which the charge is principally composed, spread out less than commonly represented. There is, however, a movement of the entering charge towards the front of the furnace, in consequence of which the ore may be observed floating down in two ridges towards that end, and gradually spreading out. Heat is taken up not only from the flame, but also from the highly-heated molten bath of the furnace, in consequence of which fusion proceeds rapidly.

Before dropping the next charge, the side-door near the hoppers is opened, and a rabble inserted to determine that no unfused material remains. It sometimes happens that some of the charge becomes agglomerated, and, moving down towards the front end, escapes fusion. These "floaters," as they are termed, are moved up toward the fire-bridge by inserting a rabble at the proper doors, so that they are brought to the hottest part of the furnace and then fused; or, better, a pipe is inserted under them by which the matte just under the floater is bessemerized, and the mass melted at the increased temperature resulting. Skimming is performed every 4 hr., care being taken, however, to do it not less than 45 min. after a charge has been dropped. It takes about 15 min., the amount run out being from 45 to 50 tons. The slag is granulated in water, and is run to waste by a launder to the dump. This operation, called skimming, is rather a tapping, the rabble being used to hold back and control the speed of the outflowing stream of slag. A rabble with a 5- by 9-in. blade and a 12-ft. handle is used, and sometimes a larger one, having a 6- by 16-in. blade. The slag runs through a settling-pot, or trough, 7 ft. long, 24 in. wide

and 18 in. deep, intended as a safeguard against a possible escape of matte. The slag adhering to this pot is called pot-slag, and is sent to the blast-furnaces, since it may contain drops of matte. When as much as 200 tons of matte has accumulated in the furnace, it is tapped off, according to the amount of matte needed at the converters. Fig. 11 is a view of matte flowing into a 10-ton ladle from two reverberatory furnaces, 80 ft. distant. The compressed-air locomotive is coupled to the ladle-car, ready to move it when full. Quite commonly the matte is tapped from two furnaces simultaneously, the runner, or launder, 80 ft. long, branching to the adjacent tap-holes of the two furnaces. These tap-holes, two to each furnace, are at the hearth-level. The plugging of a tap-hole is done with a stopper-rod, or dolly, having a button of 4 in. diameter, on which is stuck a conical clay plug. This is inserted by one man, whose efforts are seconded by another, who strikes the end of the dolly with a heavy hammer, to enter the plug effectually into the tap-hole.

Coal is charged into the fire-box about once in 40 min., dropping in 3,000 lb. of coal at the four charge-openings. The level of the coal is found by feeling it with a rod introduced through the stoke-openings at the front of the fire-box. A mica-covered hole, located in the outlet-flue of the furnace, can be observed by the fireman, 125 ft. away, who regulates the firing of the fuel by observing the appearance of the flame at this hole. The coal is carried on the grate to the height of the bridge, or about 24 in., and a clinker-bed is permitted to form on the grate-bars. These bars, 2 in. square and 6 in. centers, permit a good deal of ash and unburned coal to fall through. The dropping of these materials is constant, the coke amounting to 10 per cent. of the total coal used.

About monthly it is necessary to patch or fettle the furnace near the fire-box end, where it is subjected to the greatest action of the fire. This is known by looking in at the slot of the fire-bridge where the conker-plates are observed becoming red-hot. Repairs having been determined on, the matte is allowed to accumulate in the furnace to near the level of the sill of the skimming-door. Slag is tapped and skimmed close to the matte, and then the latter is tapped dry from the furnace. Such a large amount of matte will fill a train of 16 ladles. To take care of

it, preparations have been made by having withdrawn matte just before from the blast-furnace settlers, or fore-hearths. Thus full attention can be given to this supply at the converters. At the reverberatory, half the matte is tapped off into ladles and covered; the remainder is then treated in the same way. The first three side-doors of the empty furnace are opened and about 20 tons of sand, used for fettling, thrown in against the corroded sides, as well as placed there by means of paddles. To tap the matte and to repair the furnace requires more than 8 hr., after which the doors are put up, the fires urged, and new charges added until 250 tons of ore has been put in. About once in 8 months it becomes necessary to repair the furnace fully, when it is tapped dry and allowed to cool off until it is possible for the masons to enter. This waiting may be lessened by admitting a blast of air from a fan to ventilate the portions of the furnace where the repairing has chiefly to be done—viz., the 25 or 30 ft. at the fire-box.

These reverberatory furnaces can smelt from 250 to 325 tons daily and may be termed 300-ton furnaces. They burn from 55 to 60 tons of coal in 24 hr., or 21 per cent. of the charge. It must be remembered that about 10 per cent. of this, however, is recovered in the cinders as coke. At $4.50 per ton for coal this would make $1 per ton of charge treated, to say nothing of the 600 h.p. of steam obtained from the waste gases. On account of the large body of matte and slag carried in the furnaces, the variations of temperature are less than in smaller ones. Skimming or tapping is all done at the front door, where the entering air will not cool off the furnace. Care is taken to lute up all openings, and upon the cooler front-half of the furnace-arch, to carry a layer of 3 in. of sand to retain the heat and to stop every crack. When the sand is removed at any spot upon the arch, red-hot brick shows itself. The charge being dropped near the fire-box end, the dust has a chance to settle in going the long distance of 100 ft. through the furnace, so that the loss by flue-dust is small. With a furnace-width of 19 ft., distant from the center-line of the furnace and hence far from the heat, the corrosive action is less than in a narrower furnace. The company makes its own fire-clay and silica brick, the latter being largely used in these furnaces for both walls and roof.

A type charge of these reverberatories consists of calcines, limestone (put into the charge at the McDougall roasters), and flue-dust, containing: SiO_2, 26.1; FeO, 31.3; CaO, 2.9; S, 8.1; and Cu, 9 per cent. The slag contains SiO_2, 37.8; FeO, 38.6; CaO, 4.6; Al_2O_3, 6; and Cu, 0.37 per cent.; and the loss of sulphur, by volatilization, is 33 per cent. Corresponding reverberatory-slags at Great Falls, Mont., show Cu, 0.69 per cent.; the higher copper-content, as compared with that of the Anaconda slag, being due, (1) to the lower temperature and hence less fluid slags, (2) to the better settling in the longer furnace, and (3) to the greater amount of lime contained in the Anaconda slag. The matte is tapped, through plates 24 by 24 by 2.5 in., to the matte-launders, which convey it 80 ft. to discharge it to the ladles. The temperature of the matte, as it escapes from the furnace, is 1,035° C. and where it enters the ladle 985° C.

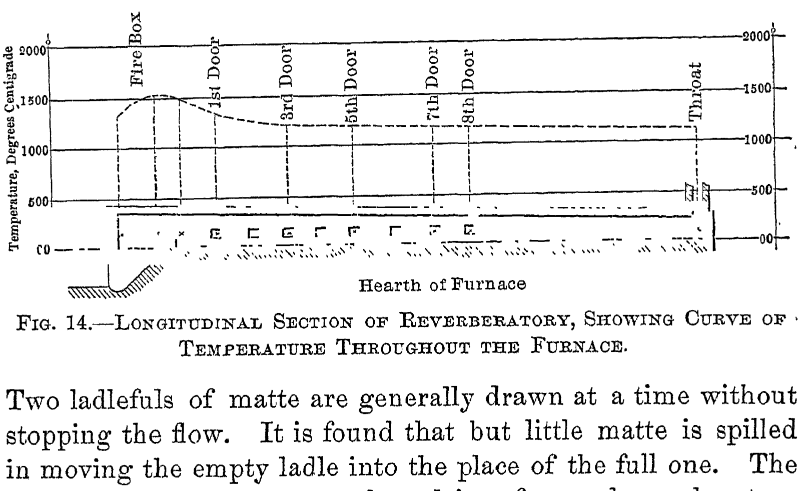

FIG. 14.—LONGITUDINAL SECTION OF REVERBERATORY, SHOWING CURVE OF TEMPERATURE THROUGHOUT THE FURNACE.

Two ladlefuls of matte are generally drawn at a time without stopping the flow. It is found that but little matte is spilled in moving the empty ladle into the place of the full one. The best point to which to roast the calcines for good reverberatory work is from 7 to 8 per cent. of sulphur.

Fig. 14 represents a longitudinal section of a reverberatory furnace, showing the variations of flame-temperature. After the third door the heat remains with but little drop to the front of the furnace. The temperature of the slag, as it issues from the furnace, is 1,120° C., and as it escapes from the separating trough, 1,060°. At the neck the escaping gases are 1,100°, and just before they enter the boiler-tubes, 950°. After passing through the boiler, the cooled gases have a temperature of from 330° to 350°. The draft increases progressively from

the fire-box, where it is 0.50 in., to the flue entering the boiler, where it is 1.25 in. of water.

As the charge drops into the furnace there ensues an active ebullition and escape of SO_2 gas, so that the escaping gases of the furnace contain 2.28 per cent. of SO_2 during the first period of 10 min. In the second and succeeding 10-min. periods, there is approximately 1 per cent. escaping. At the same time but a trace of SO_3 is to be found. Therefore, in this case, the idea that fusion with the silica of the charge produces sulphuric anhydride must be abandoned. The principal reaction is, $FeS + 3\ Fe_2O_3 + 7\ SiO_2 = 7\ FeSiO_3 + SO_2$. Of the 8 per cent. of sulphur contained in the calcines, one-third is thus evolved, the remaining two thirds going into the matte. The matte-fall is 21.5 per cent.; the matte containing 40 per cent. of copper.

XI. Converter-Plant.

The converter-floor has one end, the lining-floor, reserved for lining and drying-out the converters. On the remainder of the floor, space is reserved for the operation of 10 stands or stalls of converters, and for handling the slag and copper. The entire space of this floor is commanded by two 65-ton traveling-cranes intended for handling the converter-shells, which, newly lined, weigh 42 tons, also the slag- and metal-ladles. At one side, adjoining the lining-floor, is the lining-room, in which are the crushers and wet pans for preparing the lining or ganister for the converters. On the other side (the casting-floor) are situated two copper-refining furnaces 14 ft. by 22 ft. 8 in. and one 14 ft. by 28 ft. hearth-dimensions, and having fire-boxes 5.5 by 7 ft., or 38 sq. ft. area. The furnaces are provided with endless-chain anode-casting machines, and are filled with the molten converter-copper poured in from 5-ton metal-ladles. The converter-slag is also poured by the 5-ton ladles into a Howden slag-casting machine, which delivers it, suitably cooled by water-sprays, to hopper-bottomed railroad-cars. The converters are charged by means of 10-ton matte-ladles, which are brought in on an elevated track, and which are poured into launders leading to the mouth of the converters.

1. *Construction.*—These converters are of the barrel-type, 8 ft. in diameter by 12 ft. 6 in. in length, and each of 10 tons capacity. Their average charge is 9 tons, but when nearly eaten out they

will take a charge of from 12 to 13 tons. They are operated by hydraulic power under a pressure of 400 lb. per sq. in., actuating the usual vertical rack and sector. When freshly lined, the cavity of the body of the converter is 6 ft. long at the top, 8 ft. 4 in. long at the bottom, 4 ft. wide and 3 ft. 8 in. deep and of 80 cu. ft. capacity. In the cover this cavity tapers to the spout, 2 ft. 6 in. in diameter. The ganister, or lining, consists of siliceous ore of from 70 to 85 per cent. of silica, together with values in copper, gold and silver. To this are added, to produce the necessary adhesion, pond-slimes made by the concentrating-mill, which resemble clay, and contain 60 per cent. of SiO_2 and 2.6 per cent. of Cu. Sometimes the siliceous ore with 70 per cent. of SiO_2 will carry as much as 8 per cent. of iron, quite materially diminishing its efficiency as lining-material.

When the used-up converter-shell has been removed from its stand, the cover is taken off, and water is run into the body to cool it, so that it may be properly trimmed from projecting crusts, etc., preparatory to relining. If necessary, the 4.5-in. brick lining of the shell is repaired. The body, having been taken to the relining-stand, receives the ganister, which is dumped into it by barrow-loads in layers, each layer being rammed by means of a compressed-air rammer, resembling an Ingersoll rock-drill. This rammer, having a 20-in. stroke, and weighing two tons, is suspended from a swinging jib-crane, so that it can be moved over the surface to be rammed. The bottom is brought up to within 6 in. of the tuyere-openings. The form or mold for the cavity is now set in place and filled round with ganister, which is rammed solidly as before. Formerly the ganister was made with water into an adobe, but now it is so stiff that it takes the powerful blows of the rammer to compact it. The filling having been completed to the top of the body, the mold, made in five wedge-shaped pieces, is withdrawn. The top is lined around a form in the same way, and, to hold the lining in place, chaplets, spaced 12 in. apart, are provided, projecting from the interior surface of the shell. Finally, the top and lining are bolted together, the joint being made with adobe soft enough to compress throughout the joint. The weight of material thus added exceeds 16 tons, and the operation of lining is completed in 1.5 hr. A converter, thus lined,

will last 6 charges or pourings of copper on an average, or 2.66 tons of ganister used per charge. The lined converters are now moved over to the opposite side of the floor, where they are dried out and heated preparatory to use, firing first with wood, followed by coal. The firing is promoted by air blown in at the tuyeres from a small electrically-driven Roots blower.

2. *Operation.*—When dried, the converter is set in its stall and air turned on for a few minutes to burn up the fuel remaining in it. It is next turned down to dump out the ashes, then set in position to take its charge of matte. When the matte-ladle is ready this operation takes 5 or 10 min., the temperature of the outpouring matte being 900° C. In the older installation, the matte-launders were brought round to the front of the converter to be filled in receiving-position, after which the blast was put on as they were brought into blowing-position. At present, the launders are being changed and shortened, conducting in the matte while the converter is in blowing-position. Of course, it is necessary while charging to admit the air, in order to keep the matte out of the tuyeres. However, the short launders are preferable, as they keep themselves clear of accumulating matte, and the chances of spilling are lessened. The first charge of from 7.5 to 9 tons is all the newly-lined converter will take. The average time of blowing a straight, single charge of reverberatory-matte may be given as follows: to the first skimming for slag, 43 min.; to the second skimming, at the stage of white metal, 60 min.; and when blown to blister, 125 minutes. Under ideal conditions, and with a straight, single charge of blast-furnace matte, the following times have been attained: to the first skim, 49 min.; to the second skim, 65 min.; and to copper, 95 min. The first charge having been poured, 1,000 lb. of siliceous ore is added in a boat or scoop handled by the crane. Matte is run in on this material, which has adhered, in part at least, to the interior surface of the converter. Blowing at once proceeds, the siliceous ore saving the lining to the extent to which it has been able to satisfy the FeO set free in the blow. As the charge becomes hot, a good deal of so-called "dope," consisting of slag, matte, scrap-copper and sweepings, is put in. This is melted, lessening at the same time the intense heat of the reaction. These operations lengthen the time to 135 min. on an average, and, with 10 stalls in operation, from

65 to 90 charges are put through in 24 hr. While the average life of a converter is equal to six pourings of copper, a lining may, at times, last to as much as from 12 to 15 pourings. Forty-per cent. matte is commonly used, yet as low as 30-per cent. has been tried, but it makes less copper and takes a longer time to blow. Allowing an average of 75 charges daily of 9 tons each, 675 tons will be converted in 24 hr., producing from 40-per cent. matte 250 tons, or 500,000 lb. of metal. As successive charges are put into a converter its capacity increases by the corrosion or eating-away of the lining, the charge which it will receive increasing relatively until it attains to 12 tons.

By this time, operations have been pushed as far as is possible, and if the converter seems in danger of breaking-out, the charge may be removed in ladles and poured into other converters. It often happens that regular charges, also, are treated in this way, blowing the matte to the first or second skimming and removing it in ladles to another converter, when the original converter takes a fresh charge of 40-per cent. matte, and does not finish its own charge to copper. Spillings and cleanings from the converter sometimes accumulate beneath it. This runs down upon the ground, covering over a piece of wire rope or a chain, which has already been laid there, and to which it adheres. The crane lifts the mass by means of the imbedded chain, pulling it from beneath the converter and depositing it on a 4-wheeled bucket or car. The car is run within reach of a 10-ton traveling-crane which commands the casting-floor. The crane takes it to where it can be broken up by a drop-weight and so be made ready for recharging, either to the converter or to the blast-furnace. Whatever material, either sweepings, matte or scrap-copper of sufficiently high grade, is available, is preferably returned to the converters. Continual blowing from the converter into the hood causes an accumulation of solid material which strikes the interior of the hood, falling down to the bottom of it at the floor-level. This material, about 24 tons daily, or 5.5 per cent. of the matte treated, is cleaned out and used as "dope" for the converters. The ladle intended for the transfer of copper to the refining-furnaces is lined with finely-ground ore plastered on its interior to the depth of 4 in. This ore makes it easy to remove the skull, and also retains the heat of the copper. In the older practice, it

was customary to blow a charge to white metal and then run in another charge, the whole being then finished to blister, the operation being called "doubling." Charges are seldom doubled at present, the course preferably pursued being, either to finish the original charge to copper, or to bring it to the stage of white metal, and then distribute it to other converters.

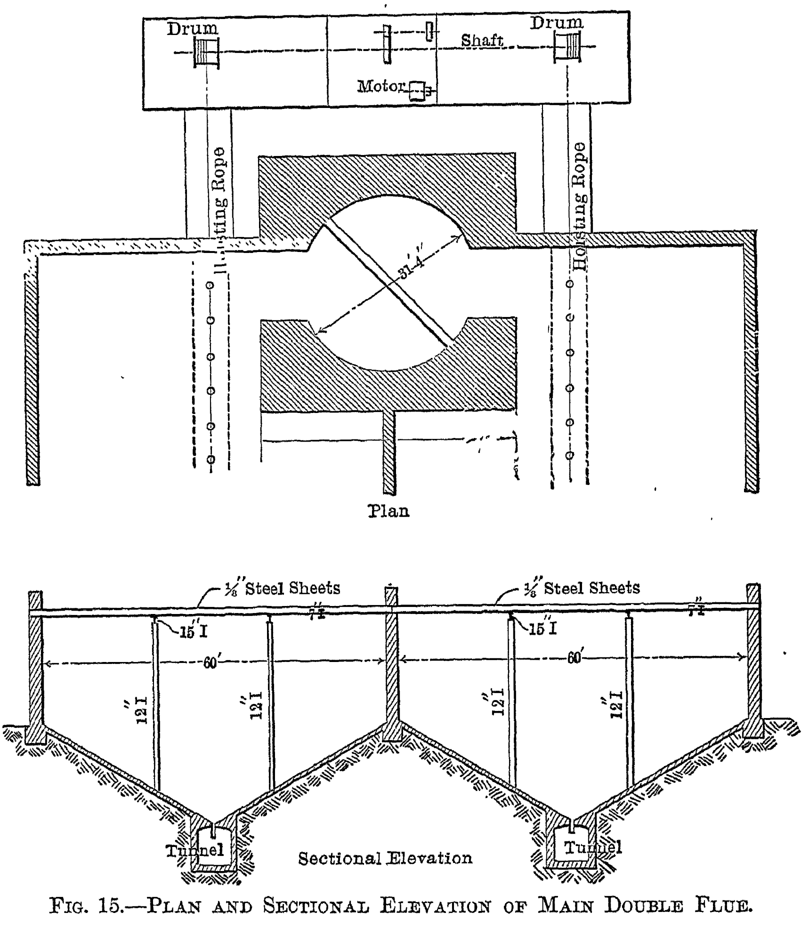

Fig. 15.—Plan and Sectional Elevation of Main Double Flue.

XII. Flues and Stacks.

The elaborate system of large flues and stacks at the works has been elsewhere described.[10] Fig. 15 shows a section and a plan of the double flue, and the location of the double hoist. At the apex at the bottom of both the single and the double

[10] *Engineering and Mining Journal,* vol. lxxvi., p. 962 (1903).

portions of the main flues there are 315 outlets, spaced every 10 ft. of its length. These outlets are controlled by slides. A covered charge-car running in the tunnel can be spotted under any one of these outlets, and by means of a canvas sleeve-connection the flue-dust can be drawn quickly into the car. The car is then lowered to the beginning of the main flue, where it is transferred to a platform-hoist by which it is raised to the arsenic-plant bins, or to the track which takes it away to be briquetted for the blast-furnace, or to the bin for supplying the reverberatory furnaces.

In spite of the pains and expense to which the company has been to insure a thorough and harmless disposal of the gases arising from the operations of the plant, a raid has been made upon it by many of the neighboring ranch-men with a view of enriching themselves by compensation for alleged damages which they still claim they incur from the fumes. These people may be divided into three classes: (1) those who realize that they are not being damaged and who, consequently, make no claim against the company; (2) those who ignorantly, but honestly, believe they are damaged; and (3) those who, while they do not honestly believe this, still make a claim for the reward that there may be in it. I have indirectly experienced the effect of such an attack upon a company, normally liberal in imparting technical knowledge of its operations, which has caused it to be reticent where it otherwise would not need to be. The advance of scientific knowledge and the cause of education is often hampered by the greed of men intent upon personal aggrandizement.

In order to show the baselessness of such a claim, the following estimate of the quantities and dilution of the gases has been made from their analyses, and from those of the ores smelted.

Gases per minute:

	Pounds.	
48 McDougall roasters, . .	25,600	
3 large, 1 small blast-furnace,	12,000	
6 reverberatory furnaces, .	6,200	
10 converters, ...	2,400	
Gases entering main flue, . .	46,200,	equaling 693,000 cu. ft. per min.

The total sulphur evolved in 24 hr. is 857 tons, which works out in the gases as 2.5 per cent. of sulphur or 5.0 per cent. SO_2 by weight (2.28 per cent. by volume).

The velocity of the air in the single main flue is 6.9 ft.; in the double flue, 3.45 ft.; and in the stack, 16.45 ft. per second.

If the outer air is still, the smoke, shot up at this velocity, dissipates itself largely in the upper regions. If the velocity of the wind is high, the smoke is rapidly torn apart and scattered. If the velocity of the wind is moderate, say 10 miles per hr., the smoke will be seen to mount at least as high again as the stack itself, thus reaching 1,000 ft. above the valley. By the time it approaches the ground it has attained a breadth of, at least, 4,000 ft. The velocity of the wind being 880 ft. per min., the new volume will be $1,000 \times 4,000 \times 880 = 3,520,000,000$ cu. ft. per min.

The dilution of the gases is, therefore, from 693,000 cu. ft. per minute to 3,520,000,000, cu. ft., or from 2.28 per cent. of SO_2 to 0.00045 per cent. of SO_2 by volume. Such a dilution is so great that, while a slight smell of gases might be distinguished, it is entirely negligible, so far as injury to animal or vegetable life is concerned.

In the various operations, an amount of sulphuric anhydride, small compared with the SO_2 gas, is also developed. This gas, speedily taking up water from the air, forms H_2SO_4. It will be recalled, in the description of the operation of the roaster, that a portion of fine limestone is added to the charge. The fine particles of this escape to the main flue and neutralize the H_2SO_4, forming calcium sulphate. Sulphates are also formed in the flue-dust with other bases, notably copper and iron. Thus no free SO_3 escapes.

The heat-operations eventually drive off all the arsenic contained in the ore, and the gases, as they cool down, deposit in the flues this element in an oxidized form. The arsenic-bearing flue-dust is treated for the recovery of its arsenic as below described.

XIII. Arsenic-Plant.

Of the total quantity of flue-dust taken from the main flue, from 60 to 80 tons daily, about 22,000 lb. of the finest portion, collected in the double flue, is taken to the arsenic-plant. At this upper portion of the main-flue, under the cooling-influence of the roof, the arsenic-fume is condensed, falling to the bottom of the flue and there mingling with the flue-dust. It is this product which is treated for the recovery of its arsenic.

The cars, in which the dust is received, are covered, and hold from 1,400 to 1,700 lb. of flue-dust, which, as it is received in them, weighs about 55 lb. to the cu. ft. The dust is dropped into the feed-hopper of the Brunton revolving-hearth roasting-furnace, of 14 ft. diameter. The charge, as the hearth revolves, is stirred by sets of blades, 33 in all, which pass through the roof of the furnace. The capacity of the furnace is 22,000 lb. daily, and there is to be another one added, doubling the output. The roasting is performed at a low heat, being at the finish of a just visible red (480° C.), the heat being kept up by a wood-fire in a fire-box adjoining the hearth. The hearth itself revolves 10 times an hour, and the dust delivered at the axis is discharged at the periphery, arsenic-free, into hopper-cars set below in a tunnel. Thus the flue-dust is periodically taken away to the reverberatory-plant. The arsenic-fume, escaping from the roaster, is conducted through a flue 240 ft. long, 18 ft. wide and 7 ft. high, to an outlet-pipe connected directly to the main flue. The arsenic-flue is interrupted every 7 ft. by baffle-walls, upon which, and upon the roof of the flue, the crude arsenic condenses and accumulates in crystalline form, containing from 85 to 92 per cent. of crude arsenious oxide (As_2O_3).

When a sufficient quantity of these crystals has accumulated in the flue, the roaster is shut down, and the condensed fume, containing As_2O_3, 85, and Cu, 4.3 per cent., with Ag, 8.1, and Au, 0.025 oz. per ton, is removed for further refining. This operation is performed in a reverberatory roasting-furnace in 6-ton charges, the arsenic being volatilized, to be again condensed in another flue, of dimensions like the former one. The residues of the operation, which remain on the hearth, are about 2 per cent. of the whole, and contain 26 per cent. of As_2O_3. The arsenic, recovered in the flue, condenses in large white crystals on the roof and walls, and contains 99.8 per cent. of As_2O_3.

The crystals are now ground in a Sturtevant burr-mill, 30 in. in diameter, and fall into a hopper having a screw-feed which delivers to a second Sturtevant mill for finer grinding. From the hopper of this second mill, having also a screw-feed, the powder is transferred and packed in barrels holding 500 lb., which are periodically jolted by power so as to settle the contents in compact form.

From the 11 tons of flue-dust put through daily, from 11 to 18 tons of arsenic are recovered monthly, being from 3.3 to 5.5 per cent. of the flue-dust. The product finds a ready market, the present price of white arsenic being from 6c. to 7c. per pound.

The workmen are protected against the poisonous effect of the fumes by the use of cotton in the nose and ears, by aspirators worn over the mouth and nose, by not working hard enough to perspire, by washing carefully after working and by anointing the exposed portions of the face with a paint of freshly precipitated ferric hydroxide rubbed on with the fingers.

XIV. Coke-Washing Plant.

The droppings from the grates of the reverberatory furnaces consist of ashes, cinders of sizes up to fist-size, coke arising from the coking of coal as it comes through the fire-bed, and sometimes pieces of coal yet unconsumed. The amount of coked material exceeds 10 per cent. of all the coal fed to the fire-boxes. These droppings fall down into a stream of water, which sweeps them away, the force of the water being sufficient to carry along even the larger pieces of slagged cinder. All passes over a grizzly, with bars spaced 1.5 in. apart, which removes the large lumps. The remainder is sluiced to the coke-washing plant, 350 ft. down the hill, and in front of the reverberatory building. Here, all is received into a V-shaped settling-tank, 20 ft. long by 6 ft. wide, divided in half by a transverse partition. This arrangement serves to unwater the materials, while from two gates at the bottom, intermittently opened (8 times per min.) by the action of the jig, a supply comes out to each of the single-compartment jigs, each 24 by 36 in., with a 9-in. overflow, a 4-in. bed and a No. 4 screen. When the supply-gate of the jig opens, a quantity of material flows out to the jig. This must be closely watched, since either the concentrates (cinders) or tailings (coke) may collect very rapidly. There is a center-discharge for the heavy portion, which is sluiced away to the dump. The washed coke next flows by a launder to a 0.25-in. mesh trommel, which takes out a great deal of dirt and ashes as well as a little fine coke. The oversize of the trommel discharges to the buckets of an inclined endless-chain elevator, which takes it to a hopper-bottom

bin located to command the track at the west end of the converter-building. Here, it is drawn off into 5-ton hopper-bottom cars to be taken to one of the stock-bins, whence it is drawn off as needed to be mixed for the briquettes.

XV. Slime-Ponds.

Upon the flat ground, below the works, are located the settling-ponds where the concentrator-slimes are collected. On the plan, Fig. 1, showing the track-system, adjoining the concentrator plant is a building 670 by 70 ft. in size, beneath which run six tracks. This building contains a series of settling-tanks, the object of which is to unwater the fine concentrates from the tables, the final overflow therefrom being conducted by launder to the slime-ponds, half a mile distant. The space occupied by these ponds, about 1,000 ft. square, is divided into six rectangular divisions, each 630 by 300 ft. The discharge of the launders is directed into the ponds, the contained water either flowing away or evaporating. Any one of the divisions may be cut out and its contents removed. To do this a Lidgerwood traveling cable-way is used, stretched from tower to tower across the ponds. The slimes are dug out by a scoop, and piled up at the edge of the division, where they drain. The towers are mounted on wheels, and may be moved like a gantry-crane, to operate over any part of the pond. Near one of the towers is placed a movable hopper-bin, having a track below, by which 50-ton hopper-bottom cars may be brought to the chute of the bin. The drained slimes are transferred by the cable-way to the bins, thence to be drawn off into the cars. These cars discharge their contents at the inclined-bottom bins of the briquette-plant or to the bin of the clay-mill at the converter-plant. The pond-slimes at the old works of the company, when dried, contain more than 5 per cent. of copper, an analysis from Hofman's paper[11] being: Cu, 4.8; Fe, 5.1; S, 7.4; SiO_2, 53.4; Al_2O_3, 20.7 per cent. and Ag, 2.8 oz. per ton, being practically a clay, still retaining a portion of the sulphides. This affords a very acceptable binder both for briquettes and for converter-lining, and all the copper is recovered. The slimes from the ponds above described contain but 2.8 per cent. of copper, and 2.6 oz. of silver per ton.

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[11] *Trans.*, xxxiv., 268 (1904).

XVI. Brick-Plant.

The brick-plant, like the foundry and machine-shops of the company, is situated in the town of Anaconda, two miles distant from the reduction-plant. The brick-plant is devoted largely to the manufacture of silica bricks, which are used extensively in the construction of the reverberatories, as well as for the crucibles and fore-hearths of the blast-furnaces.

For these bricks a local silica rock is used containing SiO_2, 96; FeO, 0.3; Al_2O_3, 2.6; and CaO, 0.6 per cent.

This material is crushed for use in brick-making. For sanding the molds, and for mortar for laying these bricks, a quartz from Dillon, Mont., is used, but, because of its friable nature, it is not suited for brick-making. It contains SiO_2, 97.5; FeO, 0.1; Al_2O_3, 1.7; and CaO, 0.2 per cent.

The quartz rock for brick-making is first crushed through a Blake crusher, and is elevated to a trommel with 0.75-in. holes, the oversize going to rolls and thence back to the trommel, while the undersize passes to a storage-bin. The material is drawn from the bin in barrows, there being added to each load a shovelful of lime-paste, so that 2 per cent. of the mass consists of CaO. This is dumped into a wet-pan, where it is mixed and ground in batches until the larger particles of the rock are no larger than will pass a 0.25-in. hole. From the pan it goes to the molding-table and is hand-molded, using Dillon sand for sanding the molds. The freshly-molded bricks are loaded on pallets, placed on iron dryer-cars, and go to the slow-drying room, where, with a good circulation of air, they are partly dried. At this stage they are re-pressed in power re-pressing brick-machines, and sent on iron dryer-cars to the hot-finishing drying-room. Here, they undergo a thorough drying-out at a high temperature given the room by an elaborate system of steam-coils set just below the floor-level. A good circulation of air through these coils carries up the heat to contact with the brick. The dried brick are then transferred to the down-draft kilns, each having a capacity of 30,000 bricks. The kilns are fired vigorously, it being found that the highest temperature attainable does not warp or fuse the brick, though, of course, the lime binding-material causes enough sintering to insure sufficient strength and compactness. A variety of sizes and kinds are made, so that, besides the regular 9- by 4.5- by 2.5-in. brick, those

of 12 and 15 in. in length are to be had. These larger sizes are particularly well adapted for the thicker walls and roof of the heavier melting-furnaces now in vogue. Table II. shows the composition of Anaconda silica brick and of other silica bricks by way of comparison. Table III. gives the analysis of Anaconda and of other clay fire-brick, some of which, because of their high alumina contents, are very resistant to basic slags.

TABLE II.—*Composition of Silica Brick.*

	SiO_2.	Al_2O_3.	Fe_2O_3.	CaO.	MgO.
	Per Cent	Per Cent.	Per Cent.	Per Cent	Per Cent.
Anaconda	97.2	....	1.1	1.0	....
Anaconda	97.0	....	2.1	0.9	....
W. H. Haws Fire Brick Co., Mt. Union, Pa.	97.2	1.4	0.2	0.76	0.35
Fayette Manufacturing Co.	96.6	0.6	0.55	1.8	0.33

TABLE III.—*Composition of Clay Fire-Brick.*

	SiO_2.	Al_2O_3.	Fe_2O_3.	CaO.	MgO.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Denniff (Anaconda)	74.2	21.7	3.4	0.0	....
Curtis No. 4 (Anaconda)	72.8	23.7	3.0	0.0	....
Acme	54.0	44.0	1.6	1.0	....
Laclede, St. Louis, Mo.	60.9	33.6	4.6	1.6	....
Evens & Howard, St. Louis, Mo.	63.3	33.6	1.9	0.7	....
Harbison & Walker, Pittsburg, Pa. (converter-tuyeres)	50.4	42.8	3.7	1.0	....
Garl Craig (Scotch)	72.4	23.4	2.4	0.0	....

Methods of Mining, Hauling, and Screening at the Mines of the Aldrich Mining Company, at Brilliant, Alabama.

BY T. H. ALDRICH, JR., BIRMINGHAM, ALA.

(London Meeting, July, 1906.)

I. INTRODUCTION.

THE Aldrich Mining Co. holds under lease from the Illinois Central R. R. Co. about 14,000 acres, in the east half of Township 12, Range 12 W., in Marion county, Alabama, and owns other lands, of which about 1,000 acres adjoin this leased tract, making a total area of about 15,000 acres, all of which is underlain by the coal-seam upon which the company is operating. This seam is nearly horizontal; and a ravine, diagonally crossing the tract, exposes the outcrop on either side for about 2 miles. The territory has been so divided that nearly the whole of it can be worked out by two collieries.

The mines at Brilliant were opened in the fall of 1898, and shipments began in the following spring. The original plan was to furnish steam-coal for locomotives to the Illinois Central R. R. Co.; and upon this expectation the investment of capital was made. The field was entirely new; and the softness of the coal at the outcrop induced the belief that it could be very cheaply mined. A contract for the delivery of a large quantity of coal to the railroad was accordingly executed. But with the advance of the entries to increasing distances from the outcrop, the coal proved so hard that it could no longer be drilled with a breast-auger; and the expense of mining became so great that the contract for cheap steam-coal had to be canceled. It became evident that the work must be done with machinery and explosives, if it could be profitably done at all.

On the other hand, the excellent quality of this coal opened, at the same time, a possible market not specially contemplated at the outset—namely, that of a first-rate coal for domestic and general uses. Since no other deposit of such coal was known to exist on the line of the Illinois Central, or any other railroad

available as customer or carrier, and since the tract controlled by the Aldrich Mining Co. could furnish for a hundred years a large annual tonnage, it was obviously worth while to make extensive experiments, in order to perfect a system of mining, haulage, dumping, etc., which would effect the minimum cost of production—including under "cost" the items (known to all mining-managers as vitally important) of maintenance, repairs, interruptions, superintendence, etc., as well as those of construction, installation, and normally requisite labor. Moreover, it was necessary to a successful system that it should yield a product suited to the special market accessible to us.

It is the purpose of this paper to state with sufficient fullness the conditions of this problem, and the solution reached under those conditions, without going into such minute details as might both transcend the limits of available space and obscure the main outline of the principles followed. It is not assumed that the practice here described would be the best for other localities and circumstances.

The situation which we had to confront comprised three principal questions, involving the best method, respectively, of mining, hauling, and dumping. These will be considered in order.

II. Mining.

The coal-seam was only 30 in. thick, and almost horizontal in position. It presented no "butts" or "faces," and the coal was so tight between the top and bottom that it could not be shot "from the solid." I have seen chunks, 6 ft. wide and 4 ft. deep, thrown, by the firing of holes containing 40 in. of powder, to such a distance from the face that a man could crawl entirely around them; and yet the coal remained as tight between top and bottom as before. There were no partings or soft streaks whatever. The roof was sometimes sandstone and sometimes slate, usually stratified sandstone. The bottom was fire-clay for about 4 in.; then gritty clay for 10 in.; then sandstone. The clay was too hard to be satisfactorily worked with any kind of machine; and sometimes the sandstone came up to the coal, with no clay between. Although the roof was excellent, and the mines were comparatively dry, mining was very difficult. We tried:

1. The long-wall system, using hand-labor with picks. This

worked well; but the necessary number of skilled miners could not be obtained.

2. The long-wall machine, which likewise gave good results, but was open to the same objection.

3. Shooting the fire-clay from the solid, and then breaking down the coal with the pick.

4. A modified long-wall system, using narrow rooms, driven by hand, and then slabbed with a long-wall machine, which was run up one side and down the other, the face being followed with a slip track.

5. Cutting the bottom with compressed-air "punchers."

6. Under-cutting the coal in the same way.

Experiments 3, 4 and 5 were semi-successful; but the only really successful one was No. 6, in which the puncher cut the coal. When the bottom only was thus cut (No. 5), there was a tendency on the part of the machine-men to keep coming up into the coal; or, perhaps, the following machine-man would cut deeper than his predecessor, thus leaving the bottom in bad condition for laying the track. Moreover, it was difficult to get the men to cut the bottom at a reasonable rate of pay; and sometimes the operation was impracticable, because the cuttings of fire-clay formed a sort of mush which could not be handled.

Commercial Considerations.

Besides the direct cost of mining a given weight of coal, special commercial considerations were involved in our problem. The hardness of our seam prevented us from competing with innumerable other producers of steam-coal, but made us the only producers of high-grade domestic coal within a large region, especially in Alabama. Our lump-coal "stocks" well, not being liable to spontaneous combustion, or to serious decrepitation. After lying for eight months in piles on the ground, it loses, in passing over the same screen, only 2 per cent. This enables us to accumulate at the mines so much of our summer output as our customers will not buy and stock for themselves. (The market-price is higher in winter, when everybody wants coal; and Alabama consumers have learned to save money by purchasing at summer-rates. But the contrary custom has long obtained in Mississippi; and such habits are hard to change.) While we would be glad to make immediate summer-deliveries

at summer-rates, we must stock our own coal somewhere, if our customers prefer not to do so; for, as all colliery-managers know, fluctuations in the labor-force and in output, due to fluctuations in market-demand, are economical evils of the worst class, injuring employers, employees and consumers alike. We have found that by stocking coal at the mines, instead of the yards where it is sold, we can save as much as 40c. per ton—partly in interest on the high freight-rates to local railroad stations, which we thus avoid paying until the coal is actually called for.

Since lump-coal is our chief profitable product, and, moreover, the size which can be stocked, as above described, with minimum loss, it is obvious that we must produce as large a proportion of lump-coal as possible, and that the size so denominated shall be as small as can be sold to consumers, or stocked without deterioration. This size has been fixed by experience to be that obtained by the use in the screen of round 2-in. holes, as further described below. The maximum production of such lump-coal requires the reduction of the machine-cutting of coal to a minimum. As already explained, we found it better to do the under-cutting in the coal than in the floor. This being the case, it was important to know which machine would give us, as a net result, the largest proportion of "lump," and the most favorable proportions of "nut" and "slack." Our experience thus far has indicated the Harrison "P. G." type as the best for our special case, on account of its ability to make a very low cut. The use of this machine gives us, of the total coal broken, about 65 per cent. lump (worth, say, $2.10 per ton), and 35 per cent. nut and slack (worth, say, $1.10 per ton). As compared with other machines which we have tried, and which yielded a smaller proportion of lump, this might save, upon the mining of 500 tons daily, for 275 days in the year, more than $30,000.

Nature of Motive Power.

The coal-cutting machines are run by compressed air, although electric power is used for haulage. Undoubtedly a large further saving could be effected by the use of electrically-transmitted power for coal-cutting, which would eliminate the serious cost of purchasing, laying, maintaining, repairing and renewing pipes, hose, hose-fittings and valves, besides reducing

the items of fuel and power-house expenses. In so small a seam, the amount of coal won for a given amount of cutting is relatively small; that of pipe-laying per ton is correspondingly large, the distance to be covered with pipes increasing very rapidly; and the pipe remaining permanently in the mine, practically never to be recovered, represents in all $10,000 per 75,000 tons of coal mined. Moreover, we have found the efficiency of the power transmitted by compressed air to be small in our practice. The coal burned at the power-house for each of our collieries is approximately 175 tons per month for each 100 tons of coal mined per day. Again, the mine-water rapidly corrodes our pipes and valves; all of the latter constantly leaking. Finally, the workmen habitually turn on the compressed air, in order to blow the smoke away, after blasting; and it is difficult to prevent this waste of power. These facts warrant the conclusion that, for our particular conditions, the electric transmission of power for the cutting of coal would effect a large saving in running-expenses.

Labor-Costs.

As already explained, mining is done by contract, each contractor receiving one machine and one entry, and delivering the coal on a large car for 75c. per ton (or on a small car, where the old system, involving switches, is still in use, at 65c. per ton). He pays his men according to a schedule fixed annually on July 1, by agreement between the company and the men, from which he is not permitted to depart. The labor-union known as "The United Mine-Workers of America," formerly organized in this locality, insisted upon the system followed in Indiana and Pennsylvania, according to which the company pays for the cutting, and the miners contract for the rest of the work—cleaning-up, timbering, track-laying, etc. This system would be advantageous for both the company and the miners, if the latter were loyal and industrious. Unfortunately, in our district, the men were like spoiled children, recklessly disregarding both their own interest and that of their employers, and working or not working, according to their whim. Under the system mentioned, we found that twice as many rooms, and nearly twice as many men, were required for a given output, and that this output was very irregular. On some

days, all the rooms would be only partly cleaned, so that none were ready for the machine. On other days, all the rooms would be ready simultaneously, while, of course, the machine was not available for all at once. It was impossible to get the men to clean up the rooms so that they could be made ready for the machine in regular succession; and consequently they were continually wrangling for precedence in this respect. Under these conditions, the output of coal per machine was on one day 30 tons; on the next, perhaps, nothing; and the average was not much more than 15 tons. Under the contract system now in force, the average output is somewhat more than 30 tons per day. A man desiring to leave one contractor and to work for another must either have the consent of the former, or give three days' notice.

The following is the present wage-schedule:

Coal-shooters, $2.50 per day; machine-runners, $0.045 per ft., or $2.50 per day; machine-scrapers, $0.03 per ft., or $1.50 per day; daily wages of drivers (one mule), $1.40; (2-mule teams), $1.65; (3-mule teams), $1.75; track-men, $2.25; miners, $2.25; outside labor, $1.25; blacksmiths, $2.35; helpers, $1.25 to $1.50; car-trimmers, $1.25; couplers (boys), $0.90; (men), $1.35; motor-runners, $1.50; dinkey-engineer, $45 per month; weighman, $40 per month,

III. Hauling.

We tried several systems of hauling, employing at first 1-ton cars, 30 in. high, which were loaded in the rooms. This required 10 in. of bottom to be taken up in the rooms, and switches to be placed at the room-necks. Later, we gained the necessary head-room from the top, instead of the bottom, because the latter, when blasted, came up badly, leaving "potholes," and not lifting in slabs or flakes. Still later, the cars were remodeled, so as to be exceptionally wide and only 18 in. high from the top of the track-rail. This made the place of loading so low that it was impossible to use any end-gate. We experimented with perhaps 100 kinds of gates, using a sloping iron-plate; a bar across the front; a chain across the front; half an end; and all manner of devices to keep the coal from coming out. All failed, because the lumps were so large, and the space in which to load them was so small. When

the coal was shot down, the center yielded large flat flakes, while the top and bottom of the seam broke with a cubical fracture, and gave a good deal of small coal, in addition to which the machine-cuttings had to be loaded and hauled out. The coal spilled badly in the gangways, causing many wrecks, much loss of coal, many crippled mules, and great expense for keeping the roads clean and the ditches open.

We tried next a large car, 6 ft. long, 6 ft. wide and 14 in. high, running on a 3-ft. gauge, with the wheels up in the car-body, protected with a cast-iron housing. This was satisfactory in many respects; but, proving clumsy to handle and to put on the track, difficult to oil, and requiring, moreover, the keeping of the track very clean, it was finally abandoned.

We then tried to increase the capacity of the car by running the bottom plank under the axle, instead of over it, and increasing the length, height and width of the car to a maximum. This experiment was also a failure, because the least little lump on the roadway would cause a wreck. Besides, for a product of 400 tons a day, it involved the daily handling of 800 small mine-cars, holding only 1,000 lb. of coal each, and thus required so many cars and mules, besides involving so much difficulty on the tipple, that a larger car was indicated as an absolute necessity. Moreover, by reason of the thinness of the seam, the rooms were worked out so fast that this system had not a sufficient radius of action without serious increase of cost.

Then we tried the old Welsh buggies, delivering the coal to cars; and finally we adopted a type of buggy made with two rib-irons, four trunnion-wheels on the sides of the irons, and five pieces of plank, with no draw-bars, bumpers or end-gates. (Later, end-gates were adopted, consisting of a plank lifting out between two guides, similar to the tail-gate of a wagon.) These buggies, which were used in the rooms only, had 14-in. wheels, were very light and cheap, could be pushed easily, and were hung so low that the bottom was on a level with the top of the rail, and the miner could load the largest lumps over the sides or ends, without laboriously building up the load. Fig. 1 shows the construction of such a buggy.

This is the system now employed. At the mouth of the room-neck, one "brushing-shot," tearing down 3 ft., is made,

and, when the buggies have been pushed to this point, the coal is transferred with a shovel from the buggies to large cars in

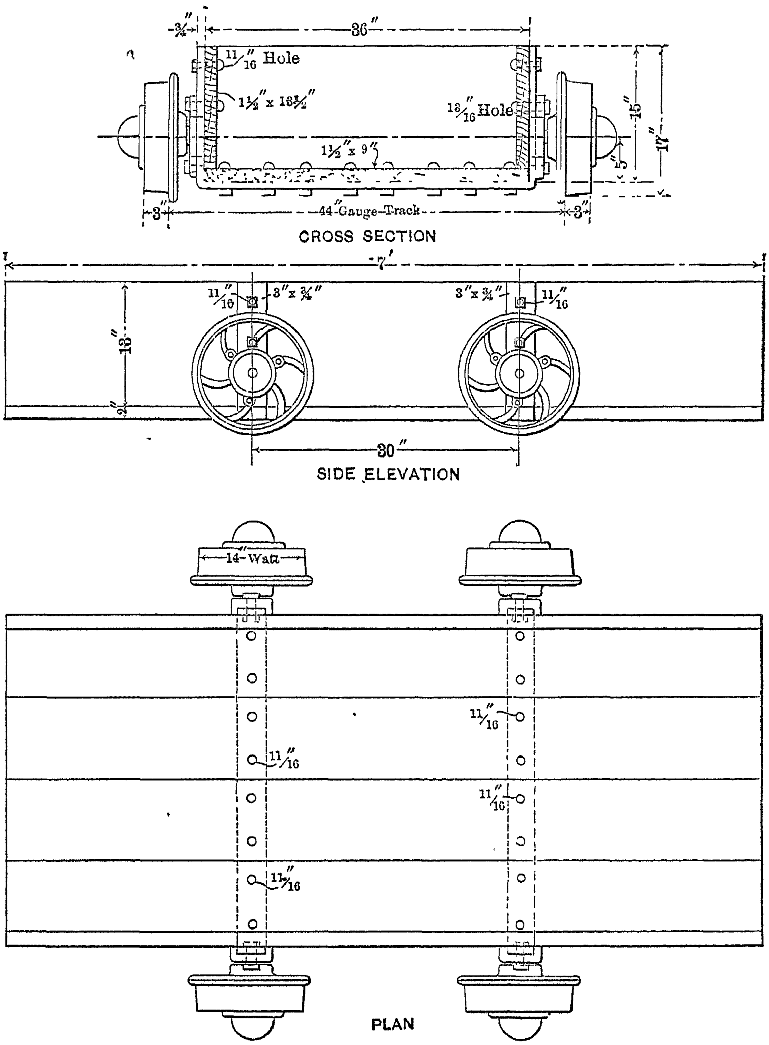

FIG. 1.—COAL-BUGGY. BUILT BY THE DECATUR CAR-WHEEL MANUFACTURING CO. FOR THE ALDRICH MINING CO.

the entry, holding 3 tons each, and having solid ends. The construction of these cars is shown in Fig. 2.

The main entry is laid with 25-lb. rails, with splice-bars and bonds; there are no room-neck switches; no coal is spilled in the roadway; no cleaning of track is necessary; and the haul-

ing is done entirely by electric power, mules being dispensed with altogether.

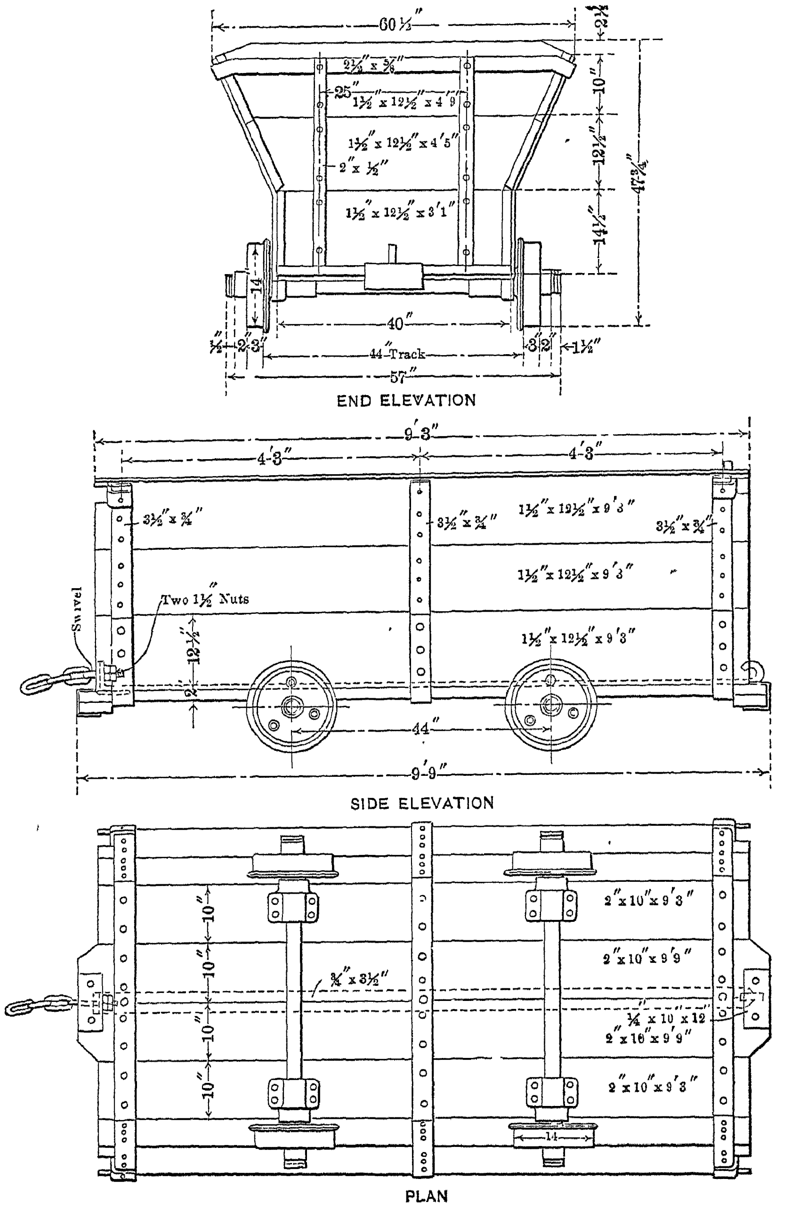

FIG. 2.—THREE-TON COAL-CAR OF THE ALDRICH MINING CO.

As already explained, mining is done by contract. Each contractor takes charge of one machine and one entry, and, as

a rule, turns out 30 tons a day, the company taking the coal from him at 75 cents per ton on the 3-ton cars. Such a contract, delivering 30 tons a day, requires 10 large cars, 5 of which are pushed into place in the morning and removed at noon, when the other 5 are put in their place. The motor makes two trips a day to each entry. On each trip, the front car is a rock-car, having side-dumps, and holding 6 tons of rock, which is about half of one "brushing-shot." With the two rock-cars furnished, the contractor can, therefore, clean up a brushing-shot each day. Since these cars are of the same size as the coal-cars, he can load them with coal if desirable.

The advantages of this system of buggies, combined with large cars and electric haulage, are as follows:

1. The absence of switches makes pipe-laying easy. Sometimes an entry will go 0.25 mile without encountering water, and suddenly there will be so much water in one place that a suction-pipe or a pump must be installed. Under the old system, this pipe might have to go under a dozen switches, at great trouble and expense.

2. No top is taken down and no bottom is taken up in the rooms, since the buggy follows the seam. Under the old system, the "yardage" paid for this work of "room-brushing" amounted to 6.1c. over and above the mining-pay of 65c. per ton, making the total labor-cost 71.1c. per ton, of which the loaders got 20c. Under the present system, 5c. more is allowed to loaders, and 5c. to the contractor, making the cost of the coal on the big car 75c. as against 71.1c. per ton. Experience has shown that, besides receiving more per ton, the loaders load more tons per day. They never have to wait for cars. The large cars spill no coal on roadways. The cost of the maintenance of cars is smaller, as is also the number of cars required.

3. The smooth main track permits of high speed, thus giving the motor greater range of action, and securing a greater utilization of gangways and tracks, in proportion to the repairs inevitably required by lapse of time.

4. Drivers and mules are eliminated.

5. Many fixed charges and contingent expenses, such as mule-feed in time of strike or shut-down, are greatly reduced.

6. Per ton of capacity, the investment in cars is much smaller.

One of the old cars weighed 1,000 lb., held 1,000 lb. of coal and cost $33 made up. One of the cars now used weighs 2,000 lb., holds 6,000 lb. of coal, and cost $47. In other words, the dead weight, formerly equal to the weight of coal, is now only one-third thereof.

7. By reason of the superior design of the present cars, the cost of maintenance per car is no greater than before; and, since the number of large cars per ton of coal mined is much smaller, the actual expense of this item per ton is only one-fifteenth of what it used to be.

8. This system requires no additional excavation. For both sizes of cars, the entry must have practically the same dimensions—namely, a height of 5 ft. 4 in. above the rail and a width of 8 ft. The satisfactory operation of any such system depends, of course, upon the perfection of its details. Some of these will, therefore, be more particularly described.

Car-Wheels.

The wheels of mine-cars are subjected to exceptionally severe wear and rough usage. The cars, having neither buffers nor brakes, are jammed together or jerked apart in hauling, or violently checked by "spragging"; the tracks are not kept in perfect condition or repaired oftener than is absolutely necessary, and the wheels are consequently jolted over irregularities and bad joints, or thrown off the rails by unnoticed obstacles, and rudely jerked on again; frequently they are covered with water; the gangways are dark, and close supervision of the haulage is impracticable; it is not always easy to detect the necessity, or perform the process, of lubrication; and, finally, the employment of as little and as cheap labor as possible is required in the operation of trains. On the other hand, the direct expense of repairs to the cars, and the inconvenience and interruption thereby occasioned (which may amount to a still greater loss), render it vitally important that the car-wheels shall run as long, and require as little personal attention, as possible. The strength of the whole chain of the haulage-system is, according to the proverb, no greater than the strength of its weakest link; and hence improvements in the design of a car-wheel may be as important as much more ambitious inventions. The wheel here described therefore constitutes an

essential element in the success of the general system adopted by our company.

Fig. 3 shows the construction of this wheel, which is a solid casting, having no parts (except the oil-plugs) which can be loosened or detached. The leading purpose of the design is to secure maximum strength for the weight of metal, uniform wear of parts, and facility and certainty of lubrication.

1. By plates, bracing the sides of the rim, a proportional strength, greater than that of wheels with spokes, is secured. Spragging may be provided for by means of a wedge-shaped lug

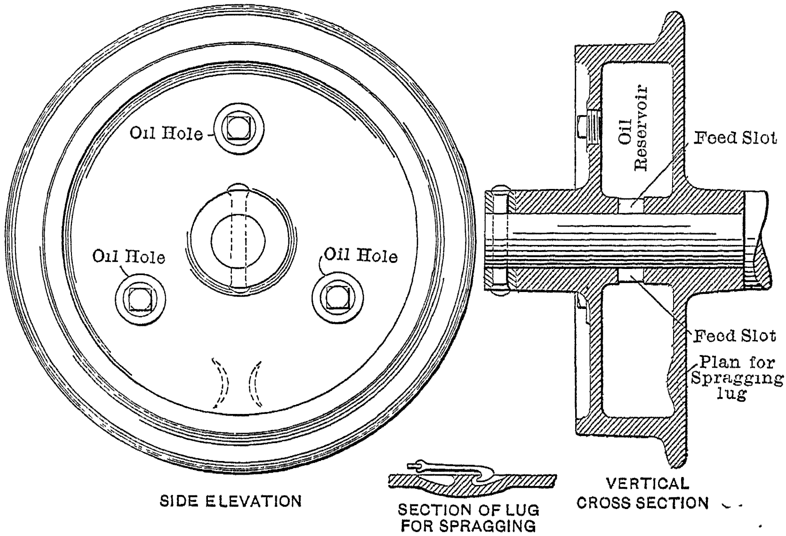

FIG. 3.—SOLID CAST-IRON WHEEL. USED BY THE ALDRICH MINING CO. UPON 3-TON MINE-CARS.

in either web, with a hook, adapted to catch and hold the wheel.

2. Uniform wear, essential to durability, is secured by an equal distribution of strain upon the bearing, and of lubricant upon the surface exposed to friction. The equal strain is obtained by means of a hub presenting an integral bearing-surface, bisected by a vertical longitudinal plane through the center of the tread of the wheel. The pressure per sq. in. is, therefore, the same at all points of the bearing-surface; and the wheel moves in true alignment with the direction of motion of the car, until it is completely worn out. The importance of this feature cannot be too much emphasized. An unequal dis-

tribution of the thrust on the bearing causes a greater wear on one side of the center, tending to a deformation, which, however minute at first, is rapidly multiplied, throwing the wheel more and more out of alignment, so that its flange not only cramps and wears the rail, but also becomes increasingly liable to "trip" at switch connections, and to wreck the car.

3. For the purpose of lubrication, the wheel is cast with webs inclosing a large oil-chamber, which extends from hub to rim, and is so large that three or, at most, four oilings per year will be sufficient. When so great a quantity of lubricant is provided, obviously a method of feeding is required which, while not liable to clog, will supply no more oil than is necessary, and will evenly distribute this supply. This is effected by one or more entirely open and unobstructed longitudinal openings or slots in the hub, leading directly from the reservoir to the axle, and so proportioned as to catch, as the wheel revolves, a suitable amount of the film of oil on the sides of the reservoir, and feed it to the center of the bearing-surface. When packing or waste is placed over or in the oil-ducts, they are liable to become clogged; and this liability calls for that closer attention to the wheel which it is desirable to render unnecessary.

In the present usual practice, the outer ends of the hubs are packed, to hold the oil in, and to protect the bearing against the entrance of grit and dirt. This involves an uneven lubrication, and also prevents early indication of the fact that the wheel has become dry. In the wheel here described, both ends of the bearing are left open, so that the oil, introduced at the center, and flowing evenly outwards in both directions, tends to prevent dirt from working in, and also shows at all times the condition of lubrication. The waste of oil, which should be prevented by a proper adjustment of the feeding-orifices, is easily noted at once; and, when the lubricant has been exhausted, the whole bearing becomes dry at the same time, the fact being patent upon the most superficial outside inspection.

Without dwelling upon other details of this design, I may mention as worthy of notice two subordinate features, which are more important in practice than they look on paper. The first is, that the openings through which the oil-reservoir is charged are so disposed that, if (as happens much oftener than

it ought to happen) a plug is carelessly *not* replaced after filling, there will be no serious loss of oil. The second is, that these plugs are made large enough to prevent their being so battered or twisted off by accidental underground shocks, such as careless wrenching, as not to be easily extracted. Any engineer who has experienced the exasperating necessity of actually drilling out a jammed plug of this kind will appreciate the value of this feature.

The wheel here described has demonstrated, by long continuous use, the superior durability, evenness of wear, and freedom from the necessity of attention, above claimed.

Car-Couplings.

The couplings of the large cars employed in this system of haulage are so designed as to permit at the tipple the tilting of each car, without breaking its connection with the preceding or the following car, and thus the connection of the train with the motor which pulls it. This problem is not difficult. It was solved in our practice by using, for the large cars, couplings consisting of three links and an eye-bolt, which is held on with jam-nuts, and the shank of which passes through the up-turned end of the draw-bar. The other end of the draw-bar is a hook; and the eye-bolt acts as a swivel.

(See Figs. 2 and 4, the latter of which shows a rock-car, like the mine-car, except that it has shutters for side-dumping. The coupling is the same for both, since the rock-car, if it should happen to be loaded with coal, would have to be dumped by tilting.)

IV. Dumping.

The tipple (Figs. 5, 6 and 7) is a cylindrical frame, rotating on trunnion-wheels, and so arranged as to permit the weighing and dumping of each car of a mine-train, without breaking the train or disconnecting the motor, and in such a manner that one man only is employed on the tipple, he attending to both weighing and dumping. Incidentally, it was necessary to provide a device which would automatically arrest and hold upon the tipple, for the purpose of dumping, each successive car of a train, while permitting the motor which hauled the train to pass over the tipple, without being thus held and overturned. This was accomplished by causing the retaining-device to en-

gage the wheel-hubs, instead of the wheel-rims, of the cars, so that the hubs of the larger wheels of the motor would pass above it, and escape its action altogether, without the need of any moving parts, or human intervention, to prevent the motor from being caught and dumped.

This system of continuous weighing and dumping (equally practicable with a steam locomotive) is operated at the mines of the Aldrich Co. with an electric locomotive as follows:

The train of loaded cars is hauled to the dump, and stops for a moment, with the motor on the tipple (the trolley wire running through it), and the first car on the weighing-scale. When this car has been weighed, the motor goes forward, bringing that car upon the tipple, where it is caught and held, while the second car, simultaneously brought upon the scale, is weighed. During the weighing of the second car, the tipple is revolved, and the first car is dumped and returned to position, without severing its connection with either the motor or the following car. The motor then advances again, and the second car is brought upon the tipple and dumped, while the third is weighed—and so on through the whole train. The interruption of train-movement required for weighing (and the simultaneous dumping of a preceding car) is very small. In fact, a skillful weigh-man, operating the tipple by means of a small lever, can weigh and dump all the cars of a train in slow motion, without really stopping them at all. After the rotation of the tipple has been thus started, the remainder of its movement is automatic; it revolves completely, comes back, and cushions itself properly in its former position, without further attention.

A small tipple of this kind, operated with the small 1,000-lb. cars, dumps regularly 7 cars, and, on a test-run, has dumped 11 cars per minute. The large tipple has a regular rate of four 3-ton cars, and has dumped, on a test-run, 7 such cars per minute. In fact, the capacity of the tipple is practically limited only by the capacity of the shaking-screen placed below it to receive the coal.

The tipple is simple and (consisting, as it does, of riveted work) very solid. The motor-car can run through the tipple as fast as 20 miles per hr. without injury to anything. It needs no attendance; does not get out of order; requires a single track only, thereby saving the cost of wooden trestles, etc., and is, on

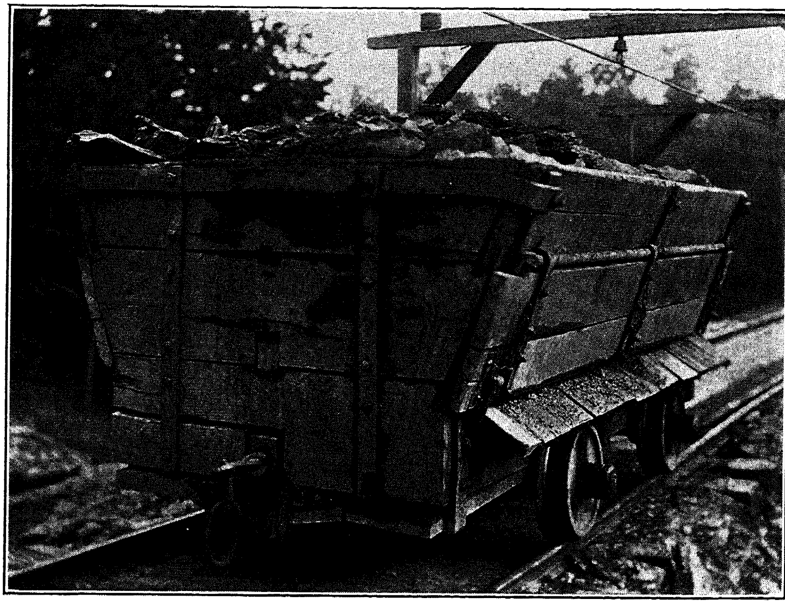

FIG. 4.—ROCK-CAR, WITH SHUTTERS FOR SIDE-DUMPING.

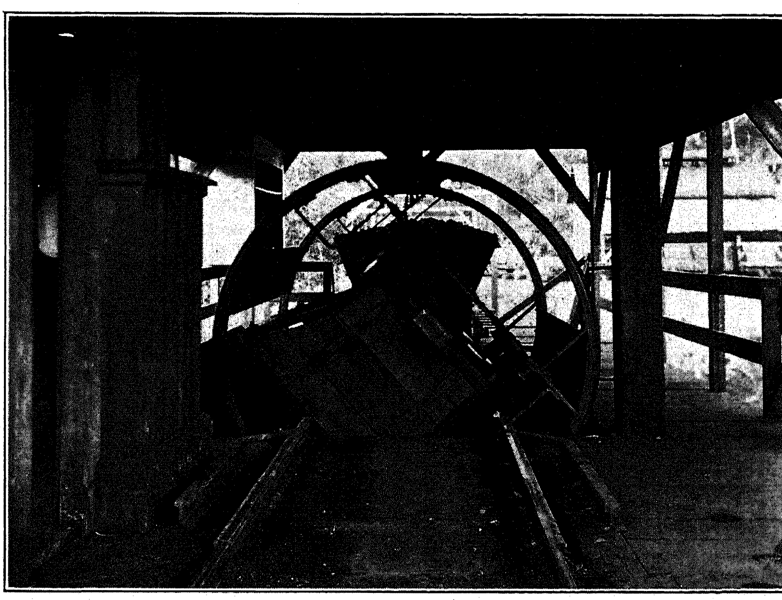

FIG. 5.—SCALE, AND TIPPLE DUMPING THE LAST CAR OF THE TRAIN.

FIG. 6.—TIPPLE, WITH SINGLE-TRACK TRESTLE, SERVING MINES ON BOTH SIDES OF THE RAVINE.

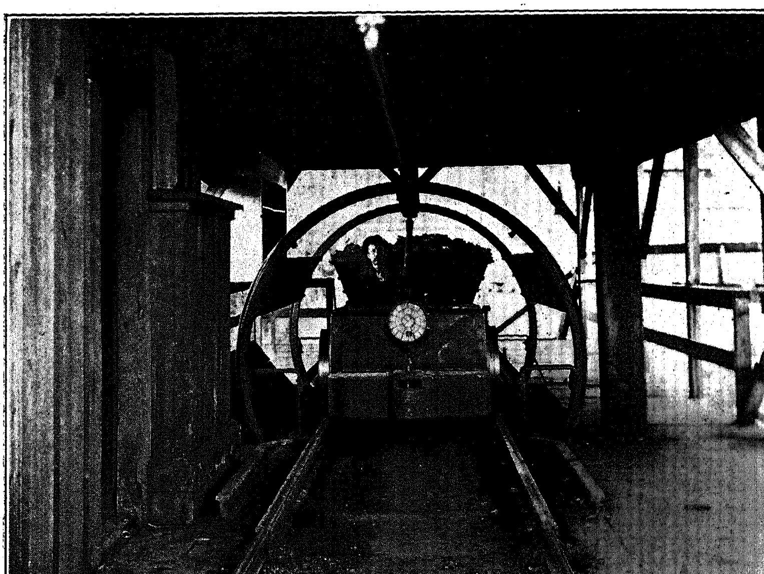

FIG. 7.—OVERHEAD TROLLEY, AND MOTOR PASSING THROUGH TIPPLE.

the whole, cheaper in construction and operation than any other automatic tipple.

The details of construction, coupling, arresting and holding of cars, and tilting-mechanism for the tipple are simple, though, of course, important with reference to the supreme need, in such an operation, of solidity, stability, safety and certainty of automatic operation. Mine-managers cannot afford to use machinery of this kind, however ingenious, which is liable to break down.

V. Screening and Washing.

As already remarked, the most advantageous limit for lump-coal has been found to be that of a 2-in. round hole in the screen. The screens upon which the coal is dumped have, therefore, holes of this size. The lump-coal thus produced sells at an average price of $2.10 per ton.

Two systems of disposing of the smaller coal passing these screens are now in use:

1. Under the first system, used at Mine No. 1, this coal is run over a screen having 0.75-in. round holes; the over-size is sold as "nut" at the average price of $1.35 per ton; and the under-size, after washing in a simple trough-washer, is sold at 50c. per ton. (This washer is very wasteful. It is estimated that 600 tons of coal per month go down stream from it. But this estimate is probably too high.)

2. Under the second system, used at Mine No. 2, all the small coal (under 2-in. diameter) is elevated and washed in jigs. The jigging is extremely simple, because there is no dirt sticking to the coal. The seam has no partings, and is absolutely clean coal from top to bottom, so that, apart from the usual slate broken from the top, the only foreign matter is fine, hard fire-clay, separated from the floor by the picks of the under-cutting machine. This fire-clay, however small in amount, is extremely objectionable, and, unless removed by washing, ruins the coal. The total weight of all impurities thus removed is about equal to the weight of the moisture in the cleaned product, so that the jigs practically furnish, by weight, as much salable coal as they receive of crude material. This product is sold without further sizing at $1.10 per ton.

The two systems may be compared as follows: For every 100 tons of coal mined, Mine No. 1 produces 53 tons of lump,

27 tons of nut, 10 tons of slack, and 10 tons is washed away by the trough-washer. At the values given above, this would net $152.80, or $1.528 per ton. In Mine No. 2, 65 tons of lump would be produced, and 35 tons of washed, nut and slack, netting the company $175.00, or $1.75 per ton.

Against the second system, must be charged a slight increase in the cost of washing, due to the difference between operation, maintenance, etc., of the jig and trough-washer plants. This is a very small item and, from the data available, cannot be figured, one man being required to operate the washer in either case, the sole difference being in power and maintenance, the item of water-supply being decidedly in favor of the jig system. After making such allowances, the No. 2 system is markedly preferable, because of the utilization of fuel, which would otherwise be lost forever. We have found the horizontal shaking-screen the most suitable for our purpose.

VI. Summary.

In the foregoing outline, the nature and extent of the economies effected as the result of our experiments have been briefly stated. The total improvement has been estimated in an official report, based upon actual practice, and dated about a year ago; since which time changes in the market-prices of coals, the labor-situation and the conditions of railway-transportation have largely increased the advantages of the new system.

The report referred to is a comparison between the old system of small cars, room-switches, mule-haulage, sizing, washing, etc., employed in Colliery No. 1, and the improved system of buggies, large cars, and electric haulage, sizing, jigging, etc., as perfected and practiced in No. 2. It shows that, under the new system, the whole plant is more cheaply constructed and maintained; that there is a greatly increased storage-capacity in mine-cars (an important item, in view of the present scanty and irregular supply of railroad-cars in this region, as everywhere else in the United States); that contractors and their workmen earn more money, while the daily output of coal per man is increased; that for a tonnage of 250 tons per day the number of workmen, outside of those employed by the contractors, is reduced from 46 to 16, a saving of $41.35 per

day; that there is an additional saving of $27.85 per day in mule-feed, switches, oil, laying of pipes and tracks, etc.; that the aggregate saving in these items of current running-expense amounts to 27.68c. per ton of coal mined, while the increase in the receipts from sales averages 22.20c. per ton, making a total of 49.88c. per ton, as the net gain of the improved system as a whole.

It is scarcely necessary to say that this result, effected in the production of a material, the selling-price of which per ton ranges from $2.10 down to $0.50, and has averaged under the old system about $1.50, and, under the new, about $1.75, is important enough to make all the difference between business success and failure, and therefore justifies the costly experiments and the radical innovations involved in the adoption of the improved system above described.

The Kurzwernhart Gas-Saving Process.

BY JOSEPH HARTSHORNE, POTTSTOWN, PA.

(Bethlehem Meeting, February, 1906.)

EVER since the introduction of the Siemens regenerative furnace, it has been recognized that a certain amount of gas is lost each time the furnace-action is reversed. This loss comes, first, from the gas which passes through the free connection between the stack and the gas-main, which exists during the time the reversing-valve is turning over; and, secondly, from the gas which is in the chamber and flues, from the hearth to the reversing-valve, and which flows back into the stack when the valve is reversed. The amount lost each time is, of course, small; but, since a furnace is reversed from four to six times every hour for, say, 300 days in the year, the aggregate is large enough to be well worth attention. The estimate has been published[1] that, in Germany alone, the loss from the first of the above causes amounts to 500,000 marks per year. The loss from the second cause is, of course, much greater.

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[1] *Stahl und Eisen*, vol. 23, p. 335 (1903).

A process for saving this entire loss, in use at Teplitz, Bohemia, for about three years, would seem to be of value even in this country, where coal is cheaper and where the small economies are not watched as closely as they are in Europe. This process, invented by Mr. Adelbert Kurzwernhart, late Technical Director of the Teplitzer Walzwerk, has been worked out with thoroughness. The loss from each cause can be calculated very closely for any particular furnace, and the results to be expected from the process ascertained. It can easily be decided, therefore, whether it is worth while to adopt it under any given conditions.

The amount of chamber-gas lost depends upon: (1) the cubic contents of the chamber, ports and flues upon one side of the furnace, up to the reversing-valve (deducting, of course, the volume of the regenerative brick); (2) upon the temperature of the gas in them; and (3) upon the number of reversals per hour. The amount of coal lost depends upon the number of cubic feet of gas evolved from a pound of it; and the final factor in ascertaining the money loss is, of course, the price of the coal per ton at the producer. All of these factors vary greatly among different works, and the value of the process for each particular case must be determined from the special conditions which there exist.

The temperature of the gas varies from about 200° C. at the reversing-valve to about 1,100° C. at the ports; 600° C. may be taken as a fair average for the purpose of calculation. In order to establish a fixed basis for comparison, the volume of the gas must be reduced to that at zero C., by the following formula, in which V^1 is the volume of the gas at zero C., V (= 1) its volume at the temperature of the chamber, x is the coefficient of expansion for each degree C., and t (= 600°) is the temperature of the chamber:

$$V^1 = \frac{V}{V + (x \times t)} = \frac{1}{1 + (0.003665 \times 600)} = 0.312.$$

In this country the volume of gas produced from a pound of coal ranges from about 25 cu. ft. for the lignites to about 72 cu. ft. for the best producer-coals. A fair average for the coals in general use appears to lie between 60 and 70 cu. ft. per

pound, say 66 ft., although some of the best authorities fix this average at 60. The saving in coal per year, in tons, is then found by the following formula, in which S is the net saving in tons, V is the net cubic content of the chamber, etc., a is the volume of gas, at zero C., from 1 lb. of coal, and n is the number of reversings per hour:

$$S = \frac{n \times 24 \times 300 \times V \dfrac{0.312}{a}}{2240} = 1.0028 \frac{nV}{a}.$$

For example, assume a 50-ton furnace, reversing four times per hour, and the cost of coal at \$2 per ton. The net cubic content of the flues, chamber and ports will be about 3,700 cu. feet. The loss per year will be:

$$\frac{1.0028 \times 4 \times 3700}{66} \times 2 = \$449.74.$$

If the furnace be reversed six times per hour, as is sometimes done, the loss will then be 50 per cent. greater, or \$674.61 per year. Of course, the poorer the coal in gas-producing quality, the greater will be the loss, since more of it is required to produce 1 cu. ft. of gas.

The loss through the reversing-valve depends on its area, the duration of the time of reversing, the velocity of the gas-current, and the temperature of the gas in the flue, together with the amount of gas produced by 1 lb. of coal, and the price of the latter. The loss of coal per year is found as follows, A being the area of the reverse-valve, D the velocity of the current (taken at 15 ft. per second), t the temperature of the gas, n the number of reversings per hour, and a the volume of gas, at zero C., from 1 lb. of coal, the duration of the reversing being taken at 5 seconds:

$$\frac{\dfrac{A \times D}{1 + (0.003665 \times t)} \times \dfrac{5 \times n \times 24 \times 300}{a}}{2240}$$

$$= \frac{\dfrac{A \times 15}{1 + (0.003665 \times 200)} \times \dfrac{n \times 36000}{a}}{2240} = 139.10 \frac{An}{a}.$$

For a 50-ton furnace, with a 30-in. gas-reversing valve, the area of the valve-opening is 4.9 sq. ft., the velocity of the gas-current is about 15 ft. per second, the temperature of the gas is about 200° C., the duration of reversing 5 seconds, and the number of reversings is 4 per hour. The loss per year, with coal at $2 per ton, will then be:

$$\frac{139.10 \times 4.9 \times 4 \times 2}{66} = \$82.61.$$

If the furnace be reversed 6 times per hour, the loss will be $123.92 per year. The total loss, from both causes, will be, therefore, $532 35, with 4 reversings per hour; and $798.53, with 6 reversings.

The idea of the Kurzwernhart process is to sweep into the furnace and utilize all the gas in the ports, chamber and flues, which is now lost in reversing, and, incidentally, to prevent any loss through the reversing-valve into the stack. This is accomplished by providing an auxiliary air-valve opening into the gas-flue between the furnace and the gas-valve, as near as possible to the latter, and by closing the gas-valve during reversing. The method of procedure is to close the gas-valve just before reversing, as is now quite generally the practice, and then to open the auxiliary air-valve. The air, rushing in through this opening, sweeps all the gas in the flues and chamber into the furnace, where it is burned with useful effect. When all of this gas has been burned, which is easily determined by observation through the peep-hole, the auxiliary air-valve is closed, the furnace is reversed, and the gas-valve is opened as before.

In considering this process, the possible danger of explosion, when the auxiliary air-valve is opened, at once suggests itself. The best comment on this suggestion is the fact, that, during the three years in which the process has been in use at Teplitz, no explosion has taken place. In reality, however, there is no such danger so long as the gas in the reversing-valve is above the temperature at which it will ignite. When it is above this temperature, the gas will always at once ignite along the line of contact as soon as the in-rushing air strikes it. The products of combustion then formed will act as a separating-layer between the gas and the air, and thus prevent further

mixing or combustion. Since the gas is pushed through the comparatively narrow flues and chamber with considerable velocity, there is neither time nor space for the dissipation of the separating-layer.

Little, or none, of the gas is, therefore, consumed in the chamber or flues, and there is no danger of burning out the brick-work more rapidly than usual. Experience has shown that there is no difference in the life of the brick-work whether the process be used or not. Of course, any heat that may be produced by the gas which burns in the valve-casing, or beyond it, goes into the furnace, or is absorbed by the checker-brick, and is thus utilized.

Another possible source of trouble is the leaking of the auxiliary air-valve during the ordinary working of the furnace, more especially because this valve is usually a simple flap. If there be a plenum in the gas-flue, as was usually the case formerly, the only result of such a leak would be a little loss of gas, which would detract just so much from the economy of the process. Under conditions of present practice, however, there is often a slight vacuum in the gas-flue, resulting in a small in-draft through whatever leak there may be. This vacuum should not ever exceed 0.2 in. of water, and probably does not average more than 0.16 in. (4 mm.). If the air so entering be of sufficient quantity, trouble might be caused by combustion, provided it entered the valve-casing, but no calorific loss would result, as stated above, except, possibly, from radiation.

There is no reason, however, why this supplemental air-valve should be allowed to leak, even if the most convenient form, that of a flap, be used. Such a valve can easily be constructed so that it will always close tightly, and a simple and effective form is shown in Fig. 1. The flap lies at an angle of about 60°, and is made stiff enough to resist warping under ordinary conditions. The bearing- and seating-surfaces are finished, and enough play is provided between the journal and its bearings to prevent jamming, and to allow the flap to adjust itself by its own weight. The upper part of the journal-bearing is made flat instead of round, to allow this self-adjustment to take place more freely. A valve so constructed will always seat itself tightly, and will continue in good condition for a long time. This form is in use at Teplitz, and it has never leaked any appreciable amount.

A proof of the absence of leakage at the auxiliary-valve, or elsewhere, is the fact that it is impossible to draw much of the gas from the chamber into the furnace by merely shutting off the gas-valve, and then waiting a certain length of time before reversing. In the absence of some opening to admit the air behind it, only so much of the gas will pass into the furnace as is forced out by the expansion due to the greater heat of the chamber. Moreover, should any leakage occur, it would have to be very considerable to do any harm. A slight amount of air mixed with producer-gas does not make it either explosive or

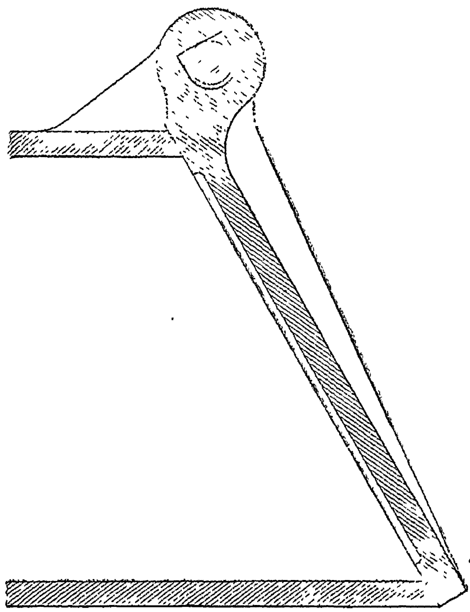

FIG. 1.—SUPPLEMENTAL AIR-VALVE.

combustible, and any small amount of flame, produced directly in front of the leak, would be extinguished by the gas itself.

Another objection which has been urged against the process is the fact that cold air is admitted to the gas-chamber, thus absorbing an undue amount of heat, which is lost when the furnace is reversed and this air passes out into the stack. The air does carry out a greater amount of heat than the gas does under present practice, but calculation shows that the loss from this cause is very small, as compared to the saving by burning the gas effectively. This is also shown by the fact that careful tests of the fuel-consumption of furnaces run for a certain

length of time with the Kurzwernhart process and for the same length of time without it, other conditions being kept as nearly similar as possible, have shown that the coal saved was 8.5 per cent. greater, by actual weight, than that which, by calculation, was to have been expected.

Although there is no danger of explosion in using air to sweep the gas into the furnace, when ordinary care is used, it can be made absolutely impossible by a modification of the process, in which the products of combustion are used for this purpose instead of air. In this case, a reservoir is provided of sufficient capacity to hold a little more than enough of these products to replace the gas in the chamber and flues. This reservoir may be built either of plates above ground, or of brick-work under ground, according to preference or convenience. During the ordinary running of the furnace, there is open communication between the smoke-flue, reservoir and chimney, which is so regulated by damper that the reservoir is kept full at all times. At the time of reversing, the main gas-valve is shut, and also the connections between the reservoir, smoke-flue and chimney. A valve, placed between the reservoir and the gas-flue, as close to the reversing-valve as possible, is then opened, and also one between the reservoir and the outer air. The air rushes into the reservoir and forces the products of combustion ahead of it into the flues and chamber, thus sweeping the gas into the furnace. As soon as all of the gas is burned, the connections between the reservoir, gas-flue and the air are closed, those between the reservoir, smoke-flue and chimney are opened, the furnace is reversed, and the main gas-flue is opened.

It will be seen, at once, that the only difference between this method, and the use of gas direct, is the substitution of a thick, extraneously produced, separating layer of products of combustion between the air and the gas for the comparatively thin layer naturally produced *in situ.* The advantages of this use of the products of combustion, instead of air direct, are that explosion is made absolutely impossible, that all trouble from a possible leak in the auxiliary air-valve is prevented, and that less heat is lost when the products of combustion pass back out of the chamber into the smoke-flue, since the temperature of these gases, when they enter the chamber, is considerably

higher than that of the outside air. As already stated, however, there is, under proper conditions, no danger of explosion when using air direct; there is no leakage of air through the auxiliary valve, when it is properly made; and the loss entailed by the use of cold air as a pushing-agent is hardly worth considering. On the other hand, the smoke-process involves a more complicated apparatus, and requires more floor-space.

The apparatus for either variety of the process can be applied to any existing form of reversing-valve. A detailed description

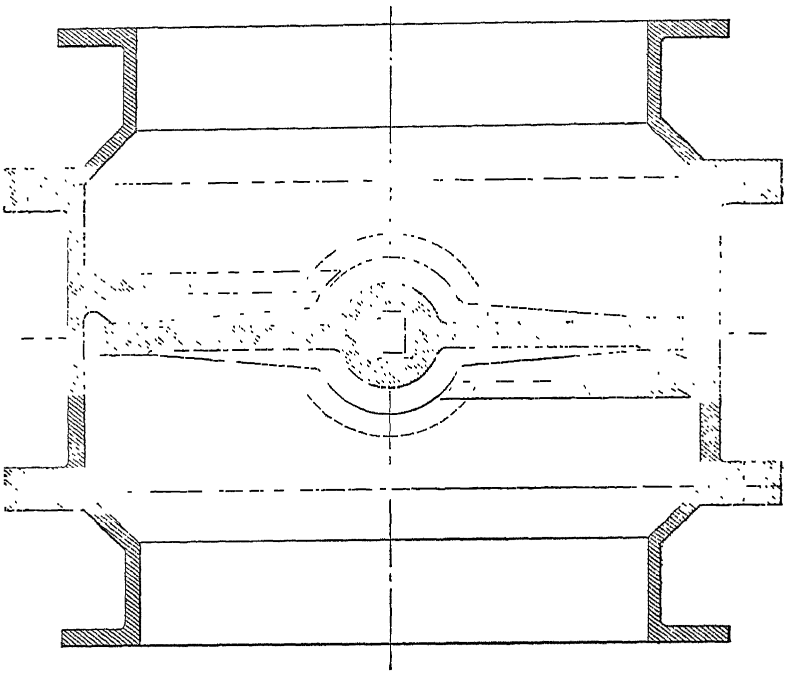

FIG. 2.—SUPPLEMENTAL GAS-VALVE.

of many of the applications of the air-process has already been published,[2] but the general principles can be sufficiently shown by means of the application to the Forter and to the Siemens reversing-valve. While with proper arrangements the main gas-valve can be used to shut off the gas during reversing, if it be preferred, or if conditions require it; yet there are some reasons which may make it seem desirable not to use the valve for this purpose; for instance, the great weight of the larger sizes, and the natural desire to avoid risking a disturbance in the re-

[2] *Stahl und Eisen*, vol. 24, pp. 937 to 944 (1904).

lation of the gas-valve to the air-valve, when once established. In such a case, a simple form of shut-off valve can be added, to be used only at the time of reversing. The question as to

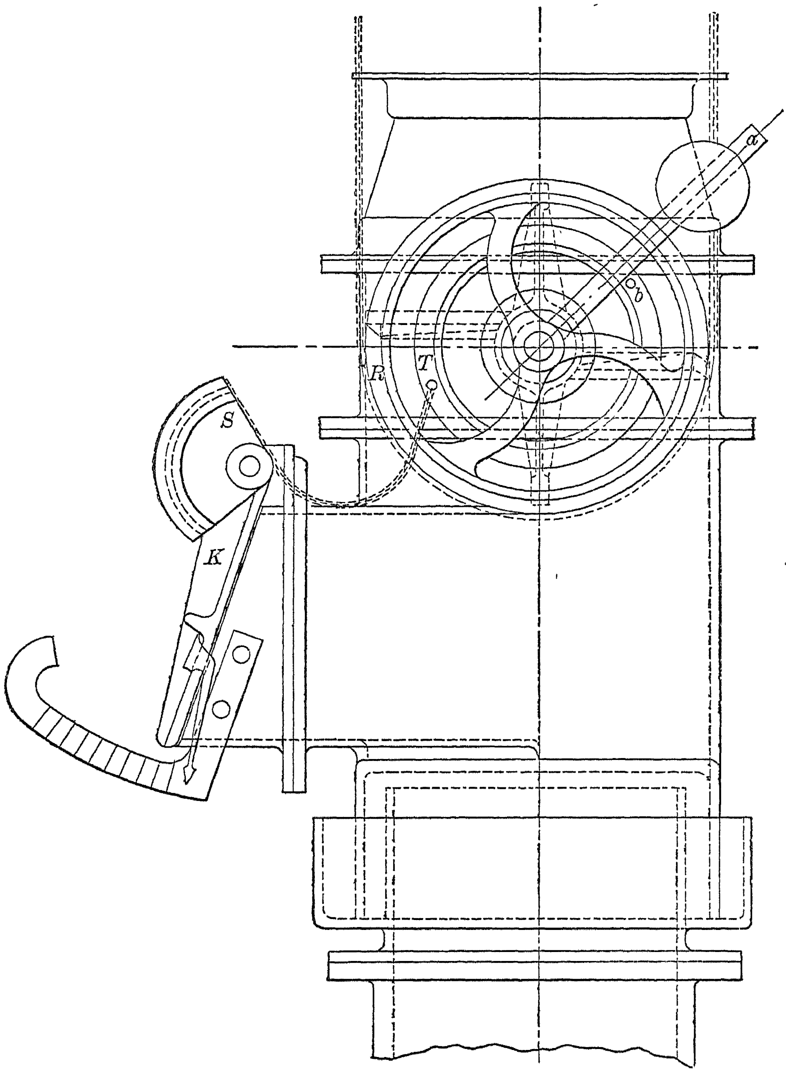

FIG. 3.—KURZWERNHART ARRANGEMENT FOR SIEMENS REVERSING-VALVE.

which method is to be followed is one of personal preference or economy, except in those cases in which there is no room to introduce the supplemental gas-valve. Fig. 2 shows a very convenient, cheap, effective and durable form of this supplemental gas-valve.

Fig. 3 shows a very convenient arrangement for the Kurzwernhart air-process, with supplemental gas-valve, as applied to the Siemens, or similar, reversing-valve. The counterweighted lever, *a*, is keyed fast to the journal on which the diaphragm of the supplemental gas-valve is fastened. The chain wheels, *R*, and *T*, which are cast in one piece, turn freely on this journal. The wheel, *T*, has a pin, *b*, projecting from it, which engages under the lever, *a*. A chain is also fastened to it, which leads to the quadrant, *S*, on the journal of the auxiliary air-valve, *K*. Another chain passes around the wheel, *R*, and also over a sprocket hand-wheel on the platform, near the lever of the reversing-valve. The supplemental gas-valve is shown in the full open position.

At the time of reversing, the hand-wheel on the platform is turned to the right, and is followed by the wheel, *R*. The lever, *a*, follows the pin, *b*, downwards, and thus closes the gas-valve tightly by means of the counter-weight. In the meantime, the slack of the chain has been taken up on the wheel, *T*, and, by continuing the motion of the hand-wheel in the same direction, the flap-valve, *K*, is opened as far as is desired. When all the gas has been burned in the furnace, the hand-wheel is turned back to the left until the air-valve, *K*, is closed. The furnace is then reversed, and the hand-wheel is turned back to its original position, thus fully opening the gas-valve. This point is fixed by a stop for the hand-wheel or lever. There must be sufficient slack in the chain leading to the air-valve to prevent the latter from opening until the gas-valve is tightly shut. The whole arrangement is simple, cheap and effective, and has successfully met all the conditions of practice.

An arrangement, to be used with the Forter valve, is shown in Fig. 4, in which the main gas-valve is also used as a shut-off valve during reversing. The stem of the gas-valve passes through the nut, *a*, which slides up and down in the frame, *B*. The opening of the gas-valve is adjusted by screw and hand-wheel, in the usual way. From the nut, *a*, projects the pin, *b*, which engages in the cam-slot of the double quadrant, *A*. Lost motion is provided for at *C*, so that the valve may be shut by means of the quadrant, no matter what may be its opening. From each end of the quadrant a chain leads to the corresponding door in the Forter valve-casing.

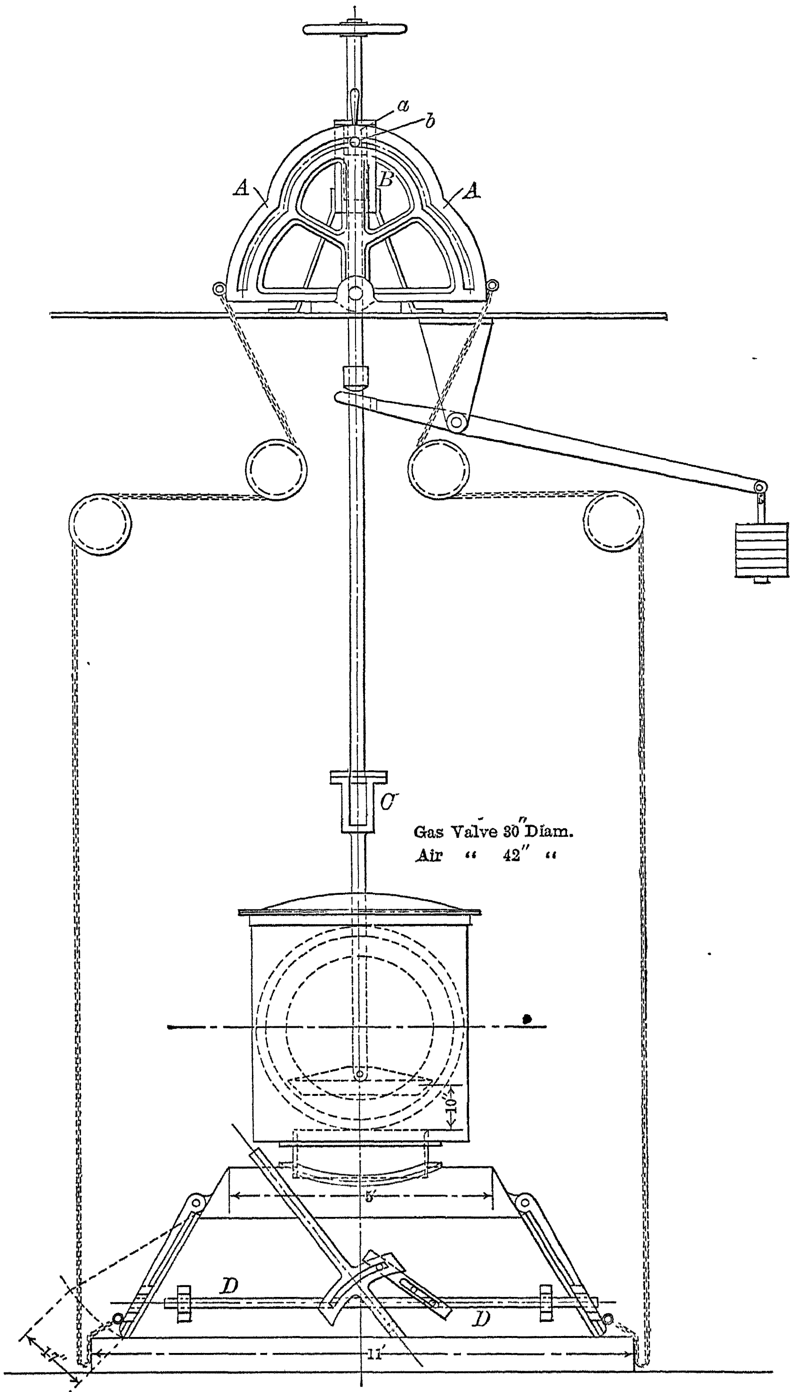

FIG. 4.—KURZWERNHART ARRANGEMENT FOR FORTER REVERSING-VALVE.

As a matter of precaution, the air is always brought into the casing over the hood, so that it will sweep all of the gas before it into the flue. If it were brought in at the other end, it would at once turn down the flue, and thus leave a pocket of dead gas at the other end of the casing, which might possibly form an explosive mixture with the air. There is small danger of this occurring, but the possibility would better be avoided. This principle of avoiding all dead ends or pockets for the gas should be applied in designing any particular arrangement, since unnecessary risk of any kind ought to be avoided. It is, therefore, necessary to use, alternately, the doors at each end.

In order to assure the opening of the proper door, and to prevent the opening of the wrong one, a bolt, *D*, is arranged on the valve-casing, operated by a cam-slot on the reversing-lever, in such a way that the door of the hood-end is free, while that of the down-comer end is locked.

When the furnace is to be reversed, the quadrant, *A*, is turned down towards the locked end of the casing. The pin, *b*, follows the slot of the quadrant, carrying with it the nut, *a*, and the gas-valve. The gas-valve is closed when the pin reaches the bottom of the semicircular part of the slot, which is then on the vertical axis of the valve-stem. The motion of the quadrant is continued in the same direction, and the air-door is opened as far as is desired. A stop should be provided for the quadrant, and the amount of opening can be adjusted by shifting the chain. Enough slack, however, must be provided in the latter to assure the closing of the gas-valve before the air-valve begins to open.

When the gas is all burned, the quadrant is raised until the air-valve is closed. The furnace is then reversed, and the quadrant is raised until the nut is again in its highest position. The main gas-valve is then open, just as it was before reversing, and its adjustment with the air-valve is not disturbed. In the case of large furnaces it will probably be necessary to counterbalance the gas-valve on account of its weight. The weight of the doors on the valve-casing will also have to be adjusted to balance the quadrant.

If it be preferred to use a supplementary gas-valve instead of the main gas-valve, for shutting off the gas during reversing, it is inserted between the latter and the casing of the reversing-

valve. The same quadrant is used, but it has no connection with the stem of the main gas-valve. The nut, *a*, becomes simply a sliding-block, to which a guided rod is fastened. This rod carries a pin, which manipulates the counterweighted lever, in the same way as does the pin, *b*, in Fig. 3.

If it be desired to use the main gas-valve as a shut-off valve during reversing, in connection with the Siemens, or similar, valve, the arrangement shown in Fig. 4 can be used, but only one-half of the double quadrant is required.

Figs. 5 and 6 show an arrangement for the use of the products of combustion as the direct agent for sweeping the gas into the furnace. In this case, the reservoir is built of brick and placed underground. *A*, *A* is the reservoir; I is the auxiliary gas-valve; *C* and *D* are the valves regulating the products of combustion and the admission of air; and *B* is the damper which controls the flow in the smoke-flue and reservoir, and which is entirely distinct from the ordinary draft-damper. During the running of the furnace, this damper is so placed as to force enough of the products of combustion through the reservoir to keep it entirely full. At this time the valves, *C* and *D*, are so set that the portion of the products of combustion which goes through the reservoir passes up through the opening II, down III, up V and down IV into the smoke-flue again. When the furnace is to be reversed, I is first shut, and the valves, *C* and *D*, are reversed. This shuts off the connections with the stack-flue at II and IV, and opens that with the outer air at VI and with the reservoir at III. The air flowing in through VI forces the products of combustion from the reservoir up through III and II into the gas-flue, just below the auxiliary gas-valve, I, thus sweeping the gas before it into the furnace. When the gas is all burned, the valves, *C* and *D*, are reversed, thus closing the connections with the stack-flue and the air, and opening those between the stack-flue and the reservoir. The furnace is then reversed, and the auxiliary gas-valve, I, is opened as before. It will be noticed that all three of the auxiliary valves are operated by a chain from one hand-wheel, and that one continuous motion will throw them all in either direction. Sufficient slack is provided in the chain to insure the shutting of the gas-valve, I, before the valves, *C* and *D*, begin to move.

If a plate-tank above ground be preferred to the underground

one, or if the space at disposal require it, the arrangement, as far as the valves are concerned, is precisely similar to that de-

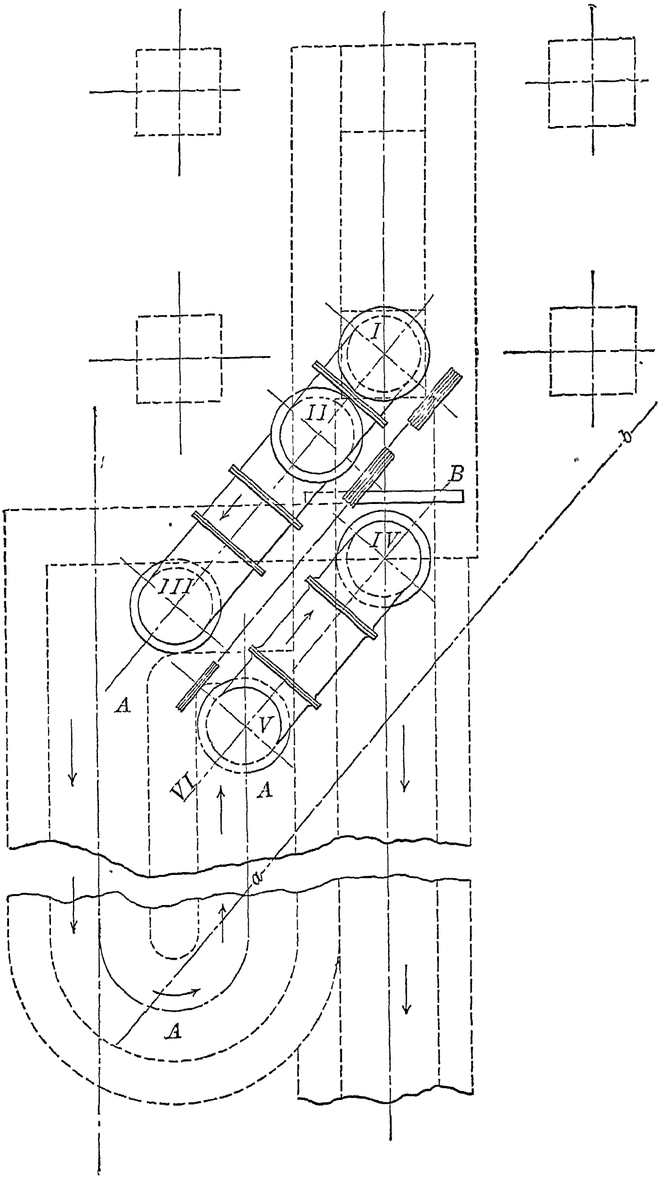

FIG. 5.—KURZWERNHART ARRANGEMENT FOR SMOKE-PROCESS. (Plan.)

scribed above, suitable connections being made through tubes. Since the products of combustion in the reservoir are considerably cooler than the gas in the chamber and flues, the former

need not be as large as the latter. A reservoir having 60 per cent. of the cubic contents of the chamber and flues will be amply sufficient.

Although I do not know that this process has been previously described in English, it has received considerable attention and

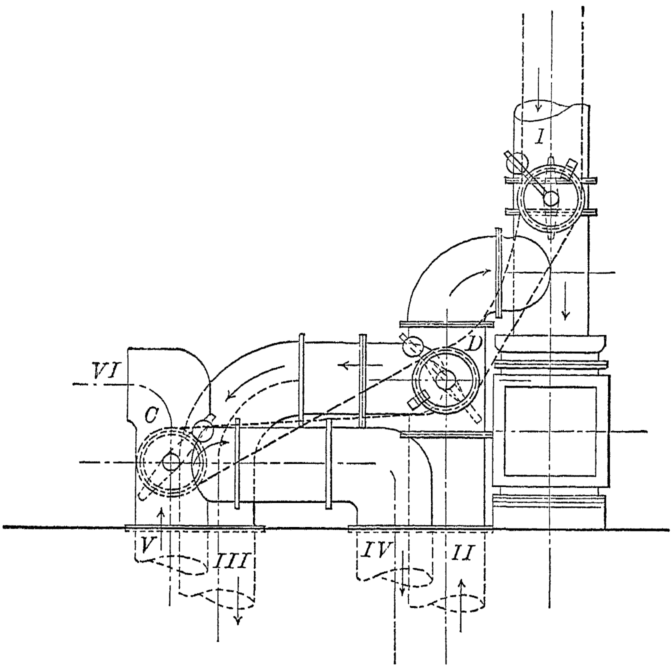

FIG. 6.—KURZWERNHART ARRANGEMENT FOR SMOKE PROCESS. (Section on a–b of Fig. 5.)

discussion on the Continent. It has been my endeavor to set before you the results obtained from the process, together with all of the objections which have been urged against it, and the answers to them which experience has developed. If it seems to any one that the aggregate saving achieved, though small in detail, is worth making, and that the process deserves investigation, all that I have hoped for has been accomplished.

The Clays of Texas.*

BY HEINRICH RIES,† ITHACA, N. Y.

(London Meeting, July, 1906.)

CONTENTS.

* Published by permission of the director of the University of Texas Mineral Survey. The chemical analyses are by O. H. Palm and S. H. Worrell, of the University of Texas.

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I. Introduction.

The facts presented in this paper, based chiefly on a reconnaissance made, during the summer of 1903, by myself and my assistant, Mr. R. C. Brooks, cover practically all that portion of Texas lying east of the 99th meridian. The work was undertaken for the University of Texas Mineral Survey, and was to have been extended over the entire State, but the sudden termination of the survey prevented even the official publication of the results already obtained.

Most of the mineral resources of Texas were fully treated by the First Geological Survey, but the clay-industry was at that time little developed, so that only a few scattered notes concerning it are to be found in the Survey reports. The extensive exploitation of the clay-deposits, however, has emphasized their commercial importance, and has also made accessible many facts of exceptional interest concerning their geologic conditions.

The erection of new plants for the utilization of clay-products is usually preceded by more or less prospecting, which, by reason of the scarcity of outcrops, and the geologic and topographic conditions of eastern and southeastern Texas, is often slow and difficult work. This is especially true in the Tertiary and Pleistocene areas, where the structural conditions much resemble those of the Atlantic coastal plain, the deposits being mostly lenticular in form, and surrounded by beds of sand.

II. General Geology of the Clay-Deposits.

The accompanying sketch-map, Fig. 1, shows the location of nearly all the deposits examined, their relation to the geology of the State, and the type of clay found in each. The geology is based on the published work of Hill, Cummins, Taff, Adams, Hayes and Kennedy, but the boundaries have unfortunately not been determined over the entire region covered by the map, and, in many areas, are only approximate. It will be seen that the clay-deposits range from Carboniferous to Pleistocene in age, the older deposits being found in the northwestern part of the area, while those of Cretaceous and Tertiary age lie to the east, southeast and south. The Pleistocene clays are found in part in a belt along the coast, and in part along many of the larger rivers, where they often underlie extensive terraces.

Table I. gives the geological succession of the clay-bearing formations of eastern Texas.

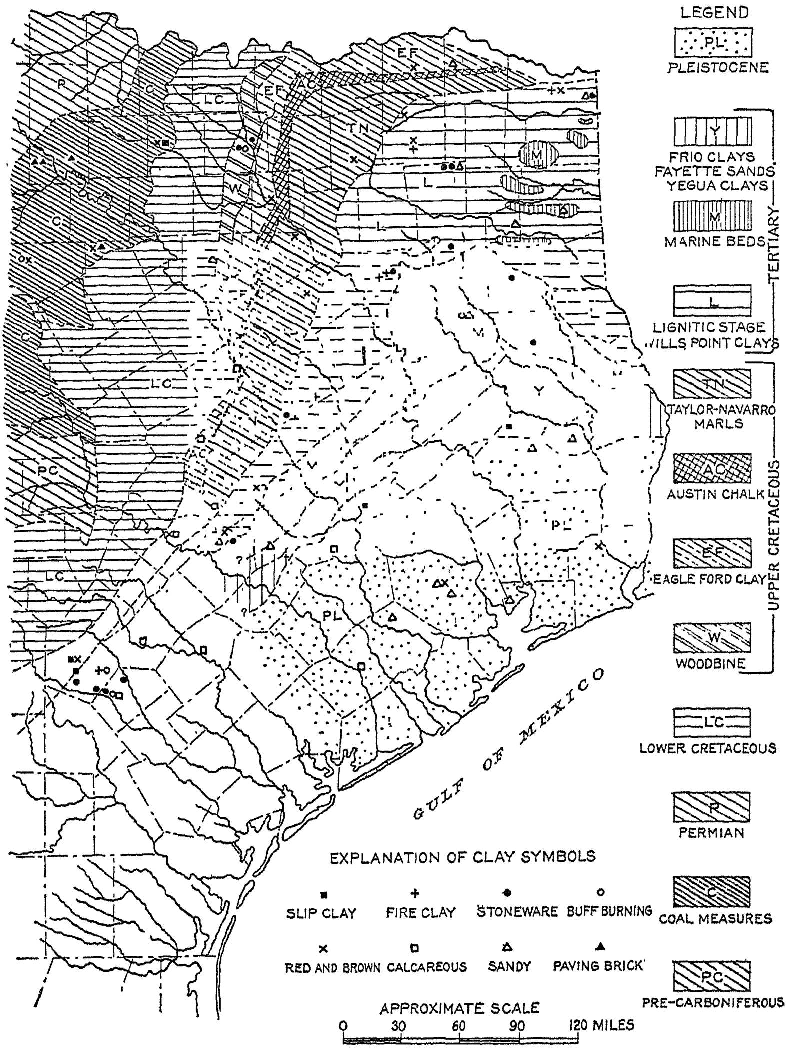

Fig. 1.—Sketch-Map of the Geology of Northeastern Texas, Showing the Location of Clay-Deposits.

III. Carboniferous Clays.

The Carboniferous rocks of northern Texas outcrop in a broad belt about 250 miles long and 45 miles in average width,[1]

[1] 22d *Annual Report, U. S. Geological Survey*, pt. iii., p. 402 (1900–01); also 1st *Annual Report*, p. 201 (1889), and 2d *Annual Report*, p. 368 (1890), *Texas Geological Survey*.

TABLE I.—*Geologic Formations of Eastern Texas.*

Age.	Period.	Formation.	Thickness.	Character of Beds.
	Pleistocene.	Beaumont clays	Feet. 25 to 400	Brown clays and sands with shells and lime pebbles.
		Columbia sands.........	50 to 200	Various colored sands above, sands and clays below.
TERTIARY.	Neocene.	Lafayette, etc...	830 to 1,355	Sands and clays, with some limestones and sandstones.
	Eocene.	Frio clays..................	160	Thinly laminated clays of various colors; many limestone concretions.
		Fayette clays.............	400	Gray sands, sandstones and clays.
		Yegua clays	1,000	Blue gypseous clays and gray sands with lignite-beds.
		Marine beds	650	Impure clays and sands, often glauconitic.
		Lignitic clays......	1,060	Sands of various colors interbedded with clays, and lignite-beds.
		Will's Point clays . . .	260	Yellowish-brown sands, laminated clays, and limestones.
CRETACEOUS.	Upper Cretaceous.	Navarro marls............	800	Calcareous clays, often sandy or glauconitic.
		Taylor marls	1,000	Calcareous clays.
		Austin chalk....	410 to 625	Limestones, sometimes chalky in character.
		Eagle Ford...............	600 or less.	Bituminous clay-shale, often gypsiferous and containing some limestone-beds.
		Woodbine..................	600 or less.	Ferruginous sandstones and clays, the latter of economic value locally.
	Lower Cretaceous.	(Contains no known clays of value, hence not differentiated.)		Limestones and marly clays.
CARBONIFEROUS.	Middle Carboniferous.	Albany.......................	0 to 1,180	Limestone and shale with thin coal-strata.
		Cisco......	800	Chiefly shale; scattered beds of limestone and sandstone as well as coal.
		Canyon.......................	800 to 930	Limestones and shales with occasional sandstones and conglomerate.
		Strawn........................	950 to 3,700	Sandstones and shales.
		Millsap.......................	1,000	Blue and black shales with occasional limestones sand and-stones; some coal.

extending from the south side of the Colorado River valley, between Lampasas and Concho counties, northward as far as the Red river in Montague county. They present a succession

of shales and sandstones, with occasional beds of limestone and coal, having a slight monoclinal dip of a few feet per mile to the west and northwest, and divisible, according to Cummins, into the five groups indicated in Table I.

Scattered through this series are a number of beds of shale of excellent quality, some of which are associated with coal-seams, and could be mined in connection with them, while others outcrop on the surface, where they are easily reached and can be economically worked. No detailed survey has been made of these shale-beds; but such as have been examined are very promising. At only three localities—viz., Thurber, Millsap and Weatherford—are they now utilized for the manufacture of clay-products; but other Carboniferous shale beds are known at Graham, Bridgeport and Cisco. None of these are yet known to be of refractory character, but they are adapted to the manufacture of paving-brick, pressed brick and pottery, and for glazing purposes.

The uniformity of the Carboniferous shale-beds is much greater than that of the Tertiary clays, and they extend over greater areas. Although as yet but little developed, these shales will probably form the basis of an important industry when their value becomes known, since they are easily accessible from the markets of Fort Worth, Dallas and many other large towns.

IV. Cretaceous Clays.

1. *Lower Cretaceous.*

The formations of this age occupy an area east and south of the Carboniferous beds. They are not utilized, nor do the stratigraphic details thus far published indicate any promising beds of clay, the formation being usually very sandy.

2. *Upper Cretaceous.*

This division carries a number of important clay-deposits, some of which are the most extensive, but unfortunately not the most valuable, in the State. The persistence in extent and the greater thickness of these deposits, as compared with the Tertiary deposits, indicate the existence of more uniform conditions during their sedimentation.

These rocks extend across Texas in a broad belt from the

Red river, north of Sherman, down to Eagle Pass, which lies in about the middle of the band; Fort Worth is on the western edge and Austin towards the southeastern border. This same series of beds also forms a belt along the Red river, the width of which narrows until it passes out of the State in the northeast corner. Since the dip is to the southeast, the older beds are found along the western edge of the belt, and the higher or younger ones on the east, where they pass below the Tertiary strata. The following members are recognized:

(*a*) *Woodbine Formation.*—This is regarded as the equivalent of the Dakota, and consists of a series of sandstones, clays and clayey sands, which often carry leaf-impressions and lignite, thereby showing their shallow-water origin. To the north it is 600 ft. thick, but thins out towards the south. While the clay-beds are usually sandy or even bituminous, they become locally pure enough, as at Denton, to be utilized for the manufacture of clay-products, although even here the beds are rarely of great extent and usually interrupted by sandy layers. The individual deposits, however, are of sufficient size to supply a fair-sized plant. In their chemical composition the clays closely resemble the stoneware clays of the lignitic stage of the Tertiary.

(*b*) *Eagle Ford Formation.*—This is one of the most extensive and thickest of the clay-bearing formations in the entire State of Texas, and consists of a series of bituminous clays which are often of more or less shaly character, with occasional thin limestone-beds or nodular septaria. The formation occupies a north-south belt, extending from a point southwest of Austin to the Red river, and then swings eastward from Bells, Grayson county, to the eastern part of Lamar county.

In the counties of Dallas, Collin and Grayson, where the formation has its greatest development, the following section has been worked out by Hill.[2]

Beginning at the bottom, there are thinly laminated deep-blue or black clays, with occasional sand-laminæ, which pass upward into less siliceous clays, containing irregular bands of thin calcareous matter and ferruginous clay nodules, as well as many lumps and flakes of selenite. The central part of the forma-

[2] 21*st Annual Report, U. S. Geological Survey*, pt. vii., p. 324 (1899–1900).

tion carries layers of arenaceous sandstone, and is then succeeded by the upper part, of blue-black clays, carrying many spherical septaria.

While the Eagle Ford clays are of great thickness and well located for working, they contain nearly all the undesirable elements that a clay may contain—namely, concretions, limestone-pebbles, gypsum-lumps, pyrite, and much organic matter. Moreover, their bituminous character and their extreme toughness cause great trouble in manipulation, and practically force the clay-worker to mold them by the dry-press process, other methods yielding a brick too dense to permit the carbon in the clay to burn off. By reason of its proximity to several large cities, the Eagle Ford clay is extensively used for brick-making, being worked at Paris, Sherman, Dallas and Waco. Fig. 2 shows a characteristic exposure of it near Dallas.

(*c*) *mustin Chalk.*—This overlies the Eagle Ford formation, and is in places an earthy limestone which carries no deposits of clay of economic value. Its intimate association with the Eagle Ford shale, however, makes it of value for admixture with the latter for the manufacture of Portland cement.

(*d*) *Taylor Marls.*—These form a somewhat broad belt underlying the "Black Waxy" region and extending parallel with the Eagle Ford shale and Austin chalk in both their N–S. and E–W. course. Their general character is that of marly clays, and in their physical and chemical properties they bear a close resemblance to Eagle Ford beds. They are not sharply separable from the next member. Fig. 3 is a pit in Taylor marls at Ferris, Texas.

(*e*) *Navarro Marls.*—These rest on the Taylor marls and represent the highest division of the Cretaceous in eastern Texas. While distinguishable with some difficulty from the Taylor marls, they are usually more sandy, and contain some glauconitic material; they also yield a black soil. To the eastward they pass under the loose beds of Tertiary material. On the geologic maps which have been published no attempt is made to separate the Taylor and Navarro marls, the two forming one belt extending from northwestern Bowie county through Red River, Delta, Hunt and Collins counties, where they turn southward and form a belt about 35 miles wide between the Austin chalk and the western border of the territory. The clays of these two

marl-formations are, on the whole, more extensively utilized than the Eagle Ford shales, and at times possess to some extent the same undesirable properties. Deposits belonging, probably, to the Navarro marls are worked at Taylor and Ferris.

3. *Comparison of Eagle Ford Clays and Taylor-Navarro Marls.*

Table II. gives the minimum, maximum and average percentages of the ingredients of these two types of clay, those of the former being the average of eight analyses, and those of the latter, of six.

TABLE II.—*Comparative Analyses of Eagle Ford Clays and Taylor-Navarro Marls, Texas.*

	Eagle Ford Clays.			Taylor-Navarro Marls.		
	Maximum.	Minimum.	Average.	Maximum.	Minimum.	Average.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
SiO_2	67.00	34.60	55.74	79.00	47.92	55.78
Al_2O_3	23.80	15.02	20.00	21.27	11.38	16.34
Fe_2O_3	6.48	1.87	4.17	8.37	2.44	4.43
CaO	21.48	tr.	4.07	12.67	0.50	7.91
MgO	2.09	0.15	1.35	2.14	0.20	1.15
K_2O	1.65	tr.	0.73	1.38	....	0.65
Na_2O	2.00	0.05	1.07	1.60	tr.	0.79
TiO_2	2.10	0.96	1.52	1.22	0.70	0.90
H_2O	7.06	5.20	6.06	6.90	3.80	4.94
CO_2	15.60	tr.	2.79	9.50	....	5.25
Organic matter	3.00	....	1.55	1.4	1.34	1.38
SO_3	3.37	....	0.79	2.00	1.12	1.52

The averages in Table II. show a close agreement, the main difference being the higher alumina-content of the Eagle Ford and the higher lime-average of the Taylor-Navarro beds (although the former showed a higher maximum percentage of lime). The difference in their composition is, however, so small that it does not make itself felt in the utilization of the clay. Both are comparatively high in organic matter—too high, in fact, to permit their being easily burned; and both run high in SO_3, due chiefly to the presence of gypsum, and, to a small extent, of pyrite. It is possible that the decomposition of the latter has been in part responsible for the formation of the gypsum, through the reaction of sulphuric acid on the calcium carbonate in the clays.

V. Tertiary Clays.

The Tertiary deposits of Texas are of both Eocene and Neocene age. The former include beds of great economic value, while the latter are usually very sandy, and carry but few argillaceous beds of good quality. While the Eocene clays are of great economic importance, the lenticular form of the deposits and the enveloping bodies of sand with which they are frequently associated render their discovery more or less difficult. The lack of outcrops and the heavy mantle of vegetation are also serious obstacles to rapid prospecting.

It appears, however, from the wide distribution of the deposits already examined, as shown in Fig. 1, that, at least in certain belts of territory mentioned below, clays may be sought with excellent chances of success. This being the case, the delineation of the boundaries of the several subdivisions of the Tertiary becomes a matter of considerable importance. Unfortunately, no published map exists, showing the boundaries of the Tertiary formation southwest of Bastrop and Robertson counties. The Tertiary beds of that portion of Texas lying east of the 99th meridian consist largely of unconsolidated materials, from coarse gravels to very fine clays, but contain also occasional beds of sandstone, limestone and lignite. The several subdivisions of the Tertiary are given in Table I., and the characters of the Eocene ones are briefly as follows:

(*a*) *Will's Point Clays.*—These beds, the basal Eocene deposits in Texas, consist usually of a stiff, laminated, yellow or bluish-green clay, with interbedded deposits of sand, as well as some calcareous beds. The clays themselves, besides being somewhat calcareous, are quite sandy, and often contain crystals of gypsum. These features collectively tend to decrease their value for the manufacturer.

(*b*) *Lignitic Clays.*—This formation, overlying the Will's Point group, outcrops through a broad irregular area, as shown on the sketch-map, Fig. 1. It comprises a series of sands and clay-beds, and often (especially near the base) carries beds of lignite, from a few inches up to 12 ft. in thickness, and, at many localities, interstratified with beds of shale and clay—the shales being sometimes semi-refractory, while the clays are non-refractory, but possess excellent plasticity. A third type,

worked at New Boston and Sulphur Springs, is a red-burning, tough, shaly clay, physically not unlike some of the Taylor-Navarro marls, but containing much less lime. A fourth and more widely distributed type is represented by lenticular deposits of grayish, highly plastic, refractory or semi-refractory clay, occurring throughout the entire lignitic belt, and already opened up in Bexar, Wilson, Limestone, Bastrop, Falls, Henderson, Smith, Wood and Bowie counties.

Table III. gives representative analyses of the clays of this group.

TABLE III.—*Analyses of Tertiary Clays, Texas.*

	I. Semi-Refractory Clays, Average of Two Analyses.	II. Red-Burning Clays, Overlying the Lignite.	III. Stoneware- and Fire-Clays.
	Per Cent.	Per Cent.	Per Cent.
SiO_2	69.33	72.9	70.65
Al_2O_3	19.38	14.7	18.14
Fe_2O_3	1.06	4.5	0.82
CaO	0.86	0.6	0.339
MgO	0.86	0.3	0.628
K_2O	tr.	1.5	0.41
Na_2O	0.08	0.7	0.55
TiO_2	1.40	1.0	1.147
H_2O	5.49	4.2	6.187

The highly siliceous character of all three classes cannot fail of notice, while II. differs from I. and III. chiefly in its percentage of ferric oxide, the excess of which seems to replace alumina. The analyses give no indication of the physical differences. All three, it will be noticed, show appreciable amounts of titanic acid. The wide distribution of these clays within the Lignitic area is shown on the map, Fig. 1.

Figs. 4 and 5 show deposits of these Lignitic clays at Saspamco and Athens. Since the heavier beds of lignite occur towards the base of the formation, the shales associated with them are to be sought next to the northwestern and northern borders of the area. Not all of the shales associated with the lignites are of good quality, some being too ferruginous and others too carbonaceous.

(*c*) *Marine Beds, or Lower Claiborne Stage.*—This formation is generally sandy or glauconitic, but carries, here and there, deposits of clay possessing some economic value, and, indeed, resembling those of the Lignitic group. The formation outcrops

in a broad hook-shaped area, mostly east and south of that underlain by the Lignitic deposits, but with some outliers in the latter, in Cass, Harrison and Gregg counties, as shown in the map, Fig. 1. Clays, presumably of the Marine stage, are worked at Rusk, Nacogdoches and Henderson. Those at Henderson, however, are so near the boundary of the Lignitic that they may possibly belong to it. If similar deposits should be hereafter found at intermediate points, the Marine beds will closely approximate in importance those of the Lignitic formation.

(*d*) *Yegua Clays, Fayette Clays and Frio Clays.*—These formations complete the deposits of Eocene age. So far as now known, they do not contain clay-beds of value.

VI. Pleistocene Clays.

Along the coast, the Pleistocene deposit forms a broad belt, extending from the southeastern border of the Tertiary to the Gulf of Mexico. The deposits are of several types, including beds of clay, sand and gravel, and, while attempts have been made to differentiate them, the subdivisions are mostly of little value to the clay-worker. Of the several groups recognized, perhaps the most important is that of the Beaumont clays (Figs. 6 and 7), which are tough, plastic, brown, blue and yellow clays, carrying irregularly distributed nodules of limestone. They underlie a broken belt extending from Calhoun to Jefferson county, and are worked for brick-making around Beaumont and Houston. At Houston, especially, the irregular distribution of the limestone nodules is very marked, the brick of one pit being full of them, while that of another may be quite free from them, and hence of much better quality. The beds are sometimes of very irregular thickness, filling depressions in a more sandy clay.

The other Pleistocene formations, underlying the broad belt bordering the Gulf, and not differentiated on the map, Fig. 1, contain scattered deposits of clay, mostly very siliceous, which are used, here and there, in the manufacture of common brick of poor quality, but have no other commercial value.

In addition to these, I should mention also the river-silts underlying the terraces along many of the larger rivers (Figs. 8 and 9), such as the Rio Grande, Colorado, Brazos, etc. These

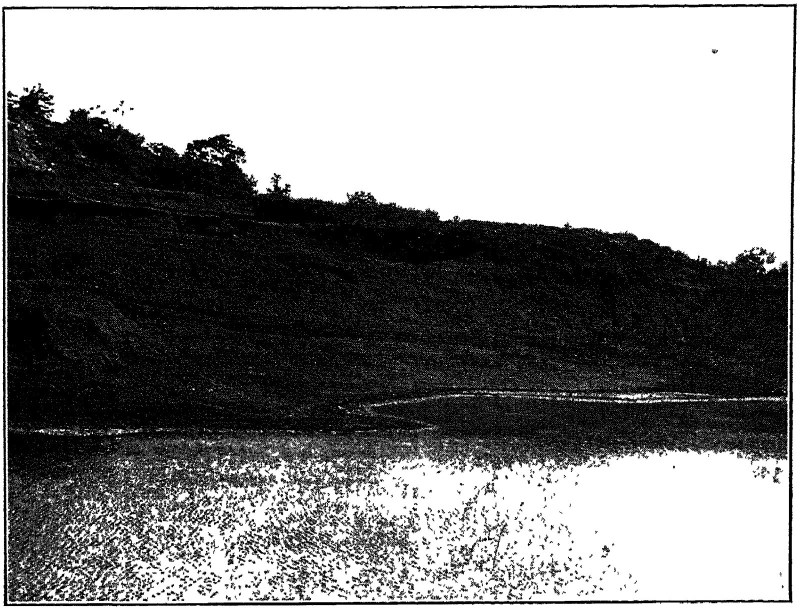

FIG. 2.—BANK OF EAGLE FORD CLAY, WEST DALLAS, TEX. (p. 526).

FIG. 3.—PIT IN TAYLOR MARLS, FERRIS, TEX. (pp. 526, 549).

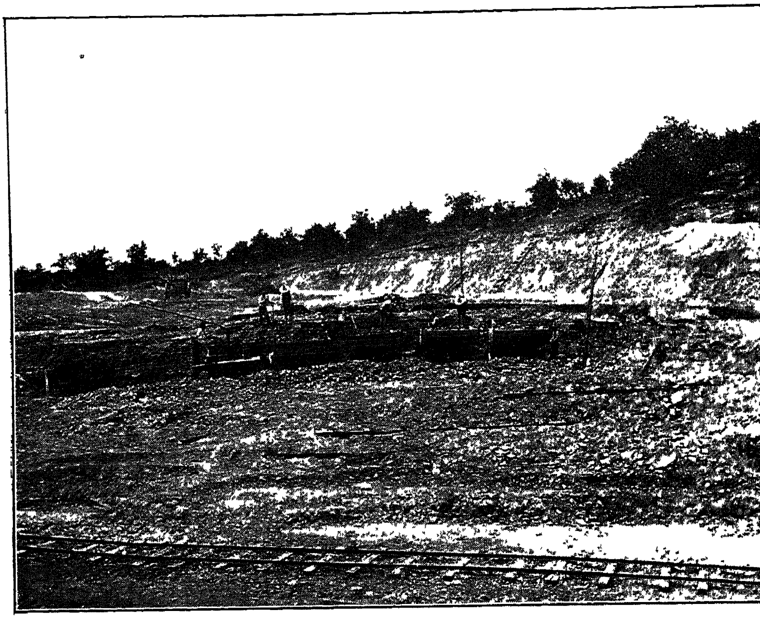

Fig. 4.—Lignitic Clay Used for Sewer-Pipe, Saspamco, Tex. (pp. 529, 541).

Fig. 5.—Lignitic Clay, Athens, Tex. (p. 529).

FIG. 6.—PIT IN BEAUMONT CLAY, COVERED WITH A LAYER OF SAND FROM 1 TO 2 FT. THICK, BEAUMONT, TEX. (p. 530).

FIG. 7.—PIT IN BEAUMONT CLAY. THE SIDES ARE LIGHT-COLORED SANDY CLAY OF COLUMBIAN (?) AGE, HOUSTON, TEX. (p. 530).

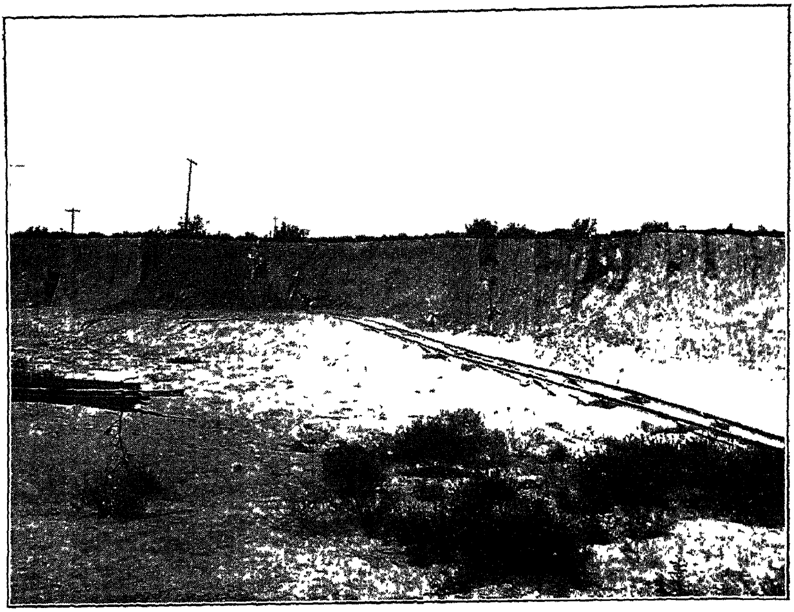

FIG. 8.—CALCAREOUS TERRACE DEPOSIT WORKED FOR COMMON BRICK CLAY LAREDO, TEX. (p. 530).

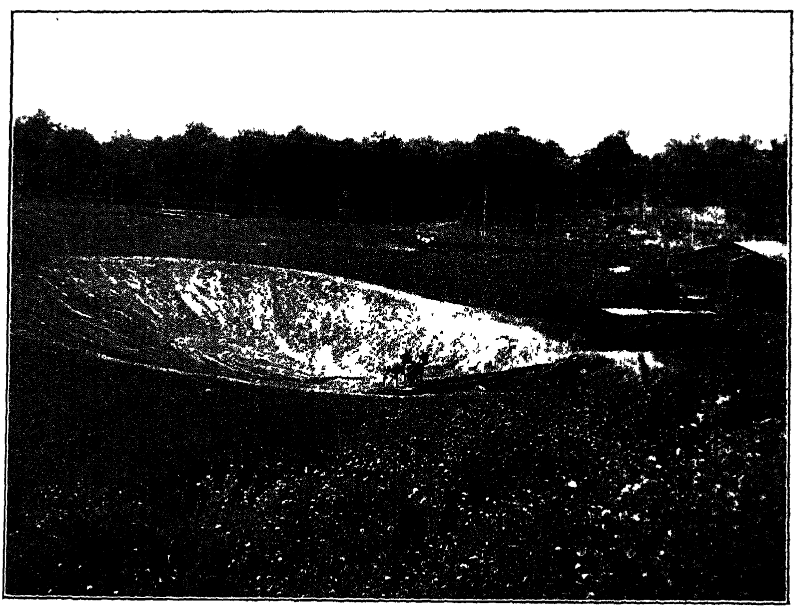

FIG. 9.—SILTY CLAY UNDERLYING TERRACE ALONG COLORADO RIVER, AUSTIN, TEX. (pp. 530, 551).

range from silty clays, through silts to clayey sands, and are nearly always highly calcareous, the calcium carbonate being present as concretions, lumps, shells, or in a finely-divided condition, and forming at times more than 50 per cent. of the material, without apparently diminishing its plasticity. They are especially well developed at Austin and Laredo.

VII. Classification of Clays.

The clays of the area under discussion have been divided into four classes, according, partly, to their physical and chemical characters, and partly to their practical applications. The names are those usually adopted in similar classifications; but in order to prevent any misinterpretation, the characters of each class will be stated. The grouping adopted is as follows:

1. Fire-clays.
2. Stoneware-clays.
3. Brick-clays.
   - (a) Buff-burning non-calcareous brick-clays.
   - (b) Red- and brown-burning brick-clays.
   - (c) Calcareous brick-clays.
   - (d) Sandy brick-clays.
   - (e) Paving-brick clays.
4. Slip-clays.

1. *Fire-Clays.*

The fire-clays include all those, the fusing-point of which is not below 1,670° C. (3,038° F., or the equivalent of Cone 27 of the Seger series). They are always low in fluxing-impurities, such as iron oxides, lime, magnesia and alkalies, contain a medium percentage of silica, and often a high percentage of alumina.

Their refractoriness increases as they approach kaolinite in composition; and it is not uncommon to divide them into fire-clays and semi-fire-clays, or into grades No. 1 and No. 2, the latter grade being frequently used for stoneware manufacture.

2. *Stoneware-Clays.*

Usually at least semi-refractory; the majority fusing not lower than 1,670° C. (3,038° F., or Cone 27 of the Seger series). They possess the excellent plasticity necessary to permit molding into jugs, pots, etc., and burn to a practically nonabsorbent body at the temperatures usually reached in stoneware-kilns.

3. *Brick-Clays.*

Used for the manufacture of common, pressed and paving-brick. Since these clays represent a wide range of material, which for each class shows more or less definite properties, the group has been subdivided as follows:

(*a*) *Buff-Burning, Non-Calcareous Brick-Clays.*—These are semi-refractory clays, which, by reason of their low percentage of iron oxide, burn to a buff color. They are frequently quite siliceous, and aside from their color-burning qualities and refractoriness, vary considerably. While their main use is for ornamental brick, they are also employed for making terra-cotta and low-grade fire-brick.

(*b*) *Red- and Brown-Burning Brick-Clays.*—Commonly of low refractoriness, due to a high content of fluxing-impurities, and burning red or brown, because of a high percentage of iron oxide. They all possess good plasticity, and burn hard and at least fairly dense, at a comparatively low heat. While the main application of these clays is for common-brick manufacture, they are also used for pressed brick.

(*c*) *Calcareous Brick-Clays.*—Those brick-clays which contain a high percentage of calcium carbonate, and in which the lime-content is commonly at least three times as great as that of the iron oxide percentage, on which account they burn buff, but differ from the sub-class (*a*) in the lower fusibility. These clays range from very plastic to very siliceous materials, yield a rather porous product, and, in the main, are adapted to little else than common brick.

(*d*) *Sandy Brick-Clays.*—Certain occurrences of red-burning clays are so highly siliceous as to warrant placing them in a group by themselves. Owing to high silica-content these clays are of low plasticity, and burn to a very porous product. Though frequently containing a high percentage of fluxes, the fusion-point is raised by the high silica-content. These clays have little value except for the commonest grades of brick.

(*e*) *Paving-Brick Clays.*—Certain plastic, fine-grained clays, of comparatively low fusibility, in which the fluxing and refractory elements are so balanced that the clays burn to a non-absorbent body, at comparatively low temperature, and attain this condition some time before they become viscous. These clays are not infrequently of shaly character.

4. *Slip-Clays.*

Clays containing so high a percentage of fluxing-impurities as to melt to a glass at the temperature at which stoneware is burned, and therefore used as natural glazes. Fineness of grain and proper chemical composition are the main factors of value in this group.

VIII. Physical and Chemical Tests of the Clays.

The economical value of a clay is most clearly determined by means of physical tests and chemical analyses, the former being the more important. In the physical tests it is necessary to determine the shrinkage in the air, as indicative of the degree of freedom from cracking in drying; the tensile strength, as showing the bonding-power; the fire-shrinkage at the temperatures at which each class of clay is likely to be burned; the absorption, to test the density to which the clay will burn; and the fusion-point. The chemical analysis gives valuable clues regarding the color-burning qualities, refractoriness, sandiness, etc. In testing the Texas clays, each sample was burned at six different temperatures; but in this paper only those results have been tabulated which were obtained at the two temperatures usually employed in burning clay in practice.

1. *Fire-Clays.*

The Texas clays which are used in the manufacture of fire-brick are refractory or, at least, semi-refractory. With the exception of a few occurrences in the Woodbine formation about Denton, these clays are confined to the Tertiary age. The deposits I examined were further restricted to the areas mapped in Fig. 1 as Lignitic and Marine—chiefly to the former. The most southerly deposit is at Adkins, Bexar county. The next occurrence northward is at Elgin, Bastrop county; and there are others in Bremond, Henderson, New Boston and Bowie counties, and two near Sulphur Springs, in Hopkins county.

At none of these points are the beds continuously traceable for any great distance; and the frequent association with sands and sandy clays indicates that the conditions over that region during their deposition must have been changeable, and

TABLE IV.—*Physical Tests and Chemical*

Locality.	Laboratory Number.	Physical Tests.						
				Cone 1. 1150° C.		Cone 9. 1310° C.		
		Average Tensile Strength.	Air-Shrinkage.	Fire-Shrinkage.	Absorption.	Fire-Shrinkage.	Absorption.	Fusion-Cone Number.
		Lb. per Sq In.	Per Cent.	Per Cent	Per Cent	Per Cent	Per Cent	
1.25 miles S of Adkins, Bexar Co...........	926	130.0	[illegible]	1.3	[illegible]	[illegible]	[illegible]	27
5 5 miles E. of Elgin, Bastrop Co............	957	186 0	9.0	4.7	6.36	5.7	0 225	30
5.5 miles E. of Elgin, Bastrop Co............	956	277.0	8.0	3.0	8.72	9.0	0.20	28
Milam Junction, Milam Co.	923	47.0	5.6	5.0	13.80	9.3	8.61	33
Bremond, Robertson Co..........................	952	43.5	4.0	0.3	13.13	.....	12 06	33
Headsville, Limestone Co.......................	954	64 0	4.0	0.6	15.97	3.0	11.92	30
Headsville, Limestone Co.......................	.....	46.0	3.3	.....	14.16	0.3	12.29	27
Athens, Henderson Co.	851	155.0	8.3	1.6	13 13	4.3	6.83	30
Athens, Henderson Co...	852	114 0	6.6	0.6	12.87	0.3	11.58	27
Malakoff, Henderson Co..........................	849	106.0	6 1	3.4	14.68	7.0	8.30	27
Malakoff, Henderson Co..........................	850	160.0	6.3	1.7	13.15	4.0	0.22	27
Sulphur Springs, Hopkins Co...................	870	68.6	4.3	2.4	13.43	4.3	8.34	31
New Boston, Bowie Co.........	861	83.0	5.1	1.6	15.35	3.4	10.54	27

that these beds are probably of irregular or somewhat lenticular form. This, however, has not prevented the occasional accumulation of bodies of considerable thickness.

One of the best sections seen was in the pit of the Athens Fire Brick Co. at Athens, where the beds dip about 5° SE. It shows, from the surface down, the following series: sand, 2 ft.; brick- and tile-clay, 8 ft.; and fire-clay, 15 ft., with sand below.

According to the Texas Survey[3] there are in this region at least five beds of fire-clay, from 2 to 12 ft. thick. A number of scattered openings have been made in them. To the northeast, in Hopkins county, the clays have been again opened up 6 miles southeast of Sulphur Springs, where the section is: mottled clay, 3 ft.; fire-clay, 12 ft.; and yellowish sandy clay, 2 ft., with sand below. This deposit has been proved over an

[3] 1st *Annual Report*, p. 36 (1889), and 2d *Annual Report*, p. 194 (1890), *Texas Geological Survey*.

Properties of Fire-Clays, Texas.

Color After Burning.	Chemical Composition.										
	SiO_2.	Al_2O_3	Fe_2O_3	CaO.	MgO.	K_2O.	Na_2O.	TiO_2.	H_2O.	Total.	Total Fluxes.
	Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Yellowish white. . .	69.70	21.50	0.40	tr.	0.50	0.30	1.00	0.12	7.10	100.62	2.20
Buff.	65 60	22.50	1.20	0 70	tr.	tr.	1.70	1.10	7.70	100.50	3.60
Buff...........	71.30	19.70	1.00	2.10	tr.	tr.	0.80	tr.	5.80	100.70	3.90
White	57.40	28 40	0.72	0.10	0 10	tr.	0.47	1 48	10 44	99.11	1.39
White	83.00	7.42	0.36	tr.	3.01	0 30	1.26	0.70	3.70	99.75	4.93
Light buff	70.82	18.90	0.40	tr.	tr.	tr.	0.50	2.10	6.80	99.52	0.90
White........................	77.40	15.70	0.70	tr.	.....	tr.	tr.	0.70	5.70	100.20	0.70
Light buff......	74.04	15 15	0.50	0.50	0.27	0.42	1.12	1.31	6.00	99.31	2 81
Buff...........................	77.29	15.29	1.59	0.33	tr.	tr.	0.12	1.18	4.75	100.55	2.04
Buff.	62.12	25 11	0.30	0.33	0.21	tr.	0 10	2.12	10.00	100.29	0.94
Buff.......	69.88	20.47	0.21	0.50	0.30	0.15	0.33	1 40	6.68	99.92	1.49
Buff...........	74.03	17 10	0.57	0.10	0.22	0.30	0.60	1.36	6.15	100.43	1.79
Buff....	73.68	17.01	0.50	0.08	1.36	tr.	0.15	1.57	6 00	100.19	2.09

area of 80 acres. The clay is slightly more refractory than that at Athens.

The deposit 6 miles south of New Boston agrees very closely in composition with that near Sulphur Springs.

Table IV. shows the results of physical tests and chemical analyses of the Texas fire-clays. Under the former head, the table gives the shrinkage and the absorption of the clay at the temperature of Seger cone No. 9 (1,310° C.), which is about the lowest heat at which the fire-bricks are likely to be burned. Its behavior at cone 1 (1,150° C.) is also given, to indicate its changes at a lower temperature.

The analyses in Table IV. show that all these clays, except one (from Milam Junction), are to be classed as siliceous, the silica-contents averaging 71.25 per cent. This somewhat impairs the refractoriness. The total fluxes range from 0.70 to 4.93 per cent., which is not excessive. The fusion-points

range from cone 27 (1,670° C.) to cone 33 (1,790° C.), 9 of the 12 tested fusing above cone 30, and two as high as cone 33. Ten of the twelve can therefore be regarded as good fire-clays. The refractoriness does not stand in direct relation to chemical composition, since the factor of texture plays an important *rôle* in some of them, as in the clays from Athens and Bremond.

The physical tests show that most of the samples have at cone 9 (1,310° C.), a low or moderate shrinkage and moderate absorption. Some, however, such as those from Elgin, Malakoff and Athens, burn too dense if used alone, and have to be mixed with more porous material.

At the present time, the Texas fire-clays are but little used. Only those at Athens have been steadily worked. This is due partly to the small demand and partly to the fact that St. Louis fire-brick are shipped into the State at a very low price. With proper management, however, there seems to be an excellent chance for developing the Texas refractories, so that they can be used not only at home, but also, to some extent at least, by Mexican consumers.

2. *Stoneware-Clays.*

The stoneware-clays are found chiefly in the Lignitic Tertiary formations, although a few are known in the Carboniferous beds, as at Rock Creek, while those around Lloyd and Denton are of Cretaceous age. Omitting, for a moment, the Carboniferous and Cretaceous occurrences from consideration, it will be seen that, beginning with the most southern locality described—viz., Strumberg, Bexar county—the stoneware-clays extend across the State in a general SW–NE. direction, the most northeasterly being at Texarkana, Bowie county (see map, Fig. 1), and all appear to be located within the areas of outcrop of the Lignitic and Marine divisions of the Eocene. There is some doubt in my mind whether those clays falling within the Marine areas as outlined on Adams's map,[4] really belong to this division, or are islands of the Lignitic, projecting through a thin layer of Marine beds. Similarly, some of those lying in the Lignitic area might belong to outliers of the Marine.

As a rule, the exposures are not large. One of the best is in

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[4] *Bulletin* No. 184, *U. S Geological Survey* (1901).

the pit of the sewer-pipe works at Saspamco (Fig. 4), which shows: (1) sandy, laminated, iron-stained surface-clay, 4 ft.; (2) chocolate clay, 8 ft.; (3) yellow ferruginous clay, 1 ft.; and (4) tough, dense chocolate clay, 7 feet.

The beds dip gently to the southeast, and the deposit can be followed along the strike for at least 1,500 ft. The clays are interstratified with sands and sandstones.

The pit 3 miles SW. of Lavernia shows: coarse yellow sand, 2 ft.; red sandy clay, 4 ft.; and dove-colored pottery-clay, 9 ft.

This, it will be seen, is quite different from the section given above. Additional sections would but serve to emphasize this irregularity. At most of the localities now worked the pits are small, and it is difficult to get any definite information regarding the extent of the individual beds without making a series of borings.

The constant feature of the deposits, however, is the character of the stoneware-clay, wherever found.

The only known occurrences of Cretaceous stoneware-clays are at Denton and Lloyd, in Denton county, where the materials used represent, according to Hill,[5] a locally pure phase of the Kiamitia clays of the Woodbine, which is Upper Cretaceous. If this be so, there is little use in hunting for them north or south of these points; and the Tertiary formations must be the main source of supply in eastern Texas. The actual conditions at Denton are not very different from those in the Tertiary belt, the section at one Denton pottery showing: yellow mottled clay (rejected), from 4 to 6 ft.; and dove-colored pottery-clay, from 2 to 3 ft.; with sand below.

The pit at Lloyd shows from 4 to 7 ft. of pottery-clay; while in the buff-burning brick-clay pit near Denton, the good clay has a total thickness of not less than 8 ft.

Table V. gives the physical properties and chemical composition of the Texas stoneware-clays, and shows them to present considerable uniformity in some respects and variation in others. They vary also in absorptive power for water (not given in the table), requiring, in mixing, from 18.7 to 34.1 per cent., with an average of 25.35 per cent. The majority show a low air-shrinkage; and, with few exceptions, the tensile

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[5] 21st *Annual Report, U. S. Geological Survey*, pt. vii., p. 295 (1899–1900).

TABLE V.—*Physical Tests and Chemical*

Locality.	Geological Age.	Laboratory Number.	Physical Tests.							
			Air-Shrinkage.	Tensile Strength (Average).	Cone 1. (1150° C.)		Cone 5. (1230° C)		Cone 9. (1310° C.)	
					Fire-Shrinkage.	Absorption.	Fire-Shrinkage.	Absorption.	Fire-Shrinkage.	Absorption.
			Per Cent.	Lb. per Sq In.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Denton, Denton Co	Woodbine formation.	845	8.6	320	2.4	5.36	2.6	3.8	4.6	2.0
Lloyd, Denton Co		913	7.0	202	1.7	7.52	3.0	6.4	3.0	4.6
Strumberg, Bexar Co.	Lignitic Tertiary.	925	5.8	183	3.0	9.46	4.7	4.26	4.7	3.7
Elmendorf, Bexar Co		808	6.8	245	3.4	8.69	4.3	6.35	4.4	2.29
Saspamco, Wilson Co		807	10.2	257	3.3	6.57	5.7	2.83	9.4	0.82
Lavernia, Wilson Co		948	7.3	204	3.3	7.8	4.6	3.6	10.66	2.1
McDade, Bastrop Co.		960	7.6	213	2.7	10.92	5.3	8.41	12.7	6.5
Denny, Falls Co		953	6.3	217	4.0	9.29	5.0	5.53	6.6	1.38
Athens, Henderson Co		853	7.0	90	4.0	12.47	6.0	8.71	10.4	1.52
Athens, Henderson Co		854	6.8	143	4.0	13.76	6.4	11.21	6.5	7.45
Tyler, Smith Co		928	6.6	224	0.7	11.81	1.7	9.16	2.0	6.51
Cornersville, Wood Co		874	5.1	66	2.7	12.15	4.3	9.14	4.7	2.27
Cornersville, Wood Co		874B	5.8	175	3.3	9.0	7.4	2.41	8.4	1.1
Winnsboro, Wood Co		872	6.4	139	2.6	11.02	4.4	8.41	7.0	2.43
Winnsboro, Wood Co		872B	7.2	163	7.0	6.08	8.9	0.21	4.6	......
Texarkana, Bowie Co		858	6.6	140	2.0	13.14	4.3	8.32	6.7	2.4
Nacogdoches, Nacogdoches Co.		855	9.6	302	2.6	5.68	4.0	3.30	5.7	4.0
Henderson, Rusk Co	Marine Tertiary.	929	6.0	89.7	4.0	12.22	5.0	9.32	6.3	6.34
Henderson, Rusk Co.		927	7.0	108	4.3	11.16	6.0	5.41	6.0	4.83

strength is excellent. In almost every case, the fire-shrinkage increases gradually, while the absorption decreases with the temperature; so that at cone 5 many of them yield a body of good density. This is notably true of the clays from Strumberg, Nacogdoches, Cornersville and Winnsboro. The temperature reached in burning by the Texas potters is not known, but is probably below the fusion-point of cone 5 (1,230° C.).

At cone 9 there is obtained, in most cases, a considerable increase in the density of the ware, without increase in the fire-shrinkage. The temperature at which the clay-particles fuse sufficiently to become steel-hard varies from cone 03 to cone 9.

Composition of Stoneware-Clays, Texas.

Fusion-Cone Number.	Color After Burning.	Chemical Composition.										Total Fluxes.
		SiO_2.	Al_2O_3.	Fe_2O_3.	CaO.	MgO.	K_2O.	Na_2O.	TiO_2.	H_2O.	Total.	
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
...	Buff	69.56	15.69	2.37	2.38	2.0	0.77	0.87	1 2	5.0	99.84	8.29
...	Deep buff	70 00	18.7	1.2	0.5	1.2	tr.	0.5	1 0	6.1	99.2	3.4
15	Buff	65.64	20.48	1.44	1.7	0.32	1.0	0.6	0.27	7.5	98.95	5.06
27	Buff	68 3	20.1	1.0	tr.	2.4	tr.	0.6	1.2	6.6	100.2	4.0
25	Buff	64.92	22.7	0.8	0.10	0.74	0.12	0.71	1.4	7.0	98.49	2.37
27	Buff	68.84	21.15	1.15	tr.	tr.	0.45	1 12	1.22	6.62	100.55	2.72
27	Deep buff	74.3	16.0	1.4	tr.	......	0.5	0.6	0.5	5.07	100.33*a*	2.5
28	Buff	68.6	20.47	0.72	tr.	0.4	1.33	0.25	1.13	6.26	99.16	2.7
28	Whitish	70.00	17.78	0.14	tr.	0.18	0.73	1.08	1 4	6.36	99.03*b*	2.13
30	Whitish	72.22	17.93	0.43	0.1	0.65	tr.	0.29	1.5	6.26	99.38	2.47
27	Buff	78.22	8.71	0.72	3.36	1.1	0.45	1.17	0.17	5.5	99.1	6.8
29-30	Buff	71.57	17.12	1.65	0.15	0.95	0.38	0.88	1.25	5.8	99.75	4.01
27	Dark buff	72.00	16.39	0.57	0.5	1.4	tr.	0.84	1.05	6.1	98.85	3.31
27	Light buff	63.3	23.4	1.6	tr.	0.3	0.9	1.0	1.2	7.6	99.3	3.8
27	Buff	72.00	17.6	0.8	tr.	0 2	1.2	tr.	1.4	5.6	98.8	2.2
27	Buff	71.2	18.0	0.6	tr.	2.0	0.9	0.3	0.7	5.8	99.5	3.8
26	Deep buff	75.33	14.73	1.1	0.05	1.61	0.64	0 1	1.27	4.5	99.33	3.5
27	Buff	69.80	15.85	1.6	3.4	0.53	0.5	1.05	0 17	6.72	99.62	7.08
27	Pink buff	67.84	21.80	1.0	tr.	tr.	0.39	1.11	1.48	7.37	100.99	2.50

a Organic matter, 3.7 per cent. *b* Organic matter, 1.36 per cent.

The average chemical composition of the Texas stoneware-clays is: SiO_2, 70.22; Al_2O_3, 18.14; Fe_2O_3, 0.99; CaO, 0.63; MgO, 1.28; Na_2O, 0.87; K_2O, 0.77; TiO_2, 1.15; H_2O, 6.80 per cent. They are therefore to be classed as siliceous clays, and their high silica-content is an influential factor in the depression of the fusion-point, which is still further lowered by the fluxing-constituents of the clay. The percentage of iron oxide is low, and hence the clays burn buff. The proportion of lime in all these clays is also small, except those which appear to belong to the Marine beds.

The fact that certain clays are here classed as stoneware-clays

does not indicate that they can be used solely for this purpose. The term is rather to be regarded as an index of certain physical qualities characteristic of stoneware-clays. While the most important use is in the manufacture of stoneware, these clays are also employed for making buff brick, floor-tile, retorts, fire-brick; in short, any kind of ware in which a fire-clay of plastic, more or less dense-burning quality, and good bonding-power is desired.

Since their value is never sufficiently high to permit their shipment to distant markets, these clays must be utilized near the deposits. The Texas stoneware-clays, therefore, await commercial development.

3. *Brick-Clays.*

The clays used for brick-manufacture in Texas can be classified on an economic basis as: (*a*) buff-burning, non-calcareous brick-clays of Carboniferous, Cretaceous or Tertiary age; (*b*) red- and brown-burning clays, of Carboniferous to Pleistocene age, for common and pressed brick; (*c*) calcareous brick clays, cream-burning, of Pleistocene age; (*d*) sandy, red-burning clays, mostly of Pleistocene age; and (*e*) paving-brick clays of Carboniferous age.

(*a*) *Buff-Burning Non-Calcareous Clays.*—These are known to occur in the Cretaceous, Lignitic and Marine Tertiary, and in the Carboniferous. The only occurrence noted in the Carboniferous (although there are doubtless others) was at Cisco, Eastland county, where the lower shale, just below the coal, is of the proper composition to yield a buff-burning brick. The material, however, would have to be mined together with the coal.

Among the Cretaceous formations there are two areas supplying material of this class. One is the district along the Rio Grande at Minera, Webb county, the other at Denton, Denton county. At Minera and Cannel, beds of carbonaceous shale, from 2 to 3 ft. thick, are found underlying both coal-seams, and, owing to the thinness of the latter, more or less of the former also has to be removed in mining the coal. For some time this shale was allowed to accumulate on the dump, but since its value was recognized it has been shipped down to Laredo. While these shales burn to a good product, their frequently high content of carbonaceous matter and pyrite renders it im-

portant that they should be well weathered before using. The economic value will continue only as long as these clays can be mined with the coal.

The clays found at Denton differ in character from the preceding, being of more desirable quality as well as better located for development. These clays occur in the Woodbine formation of the Upper Cretaceous; and the following section, from the pit of the Denton Pressed Brick Co., illustrates the mode of occurrence: blue clay, from 1 to 6 ft.; yellow-mottled clay, 5 ft.; greasy-gray clay, called "ball-clay," 4 ft.; black clay, from 2 to 3 ft.; and yellow-mottled bottom-clay, from 1 to 2 ft., with sand below.

This occurrence presents a series of beds, more or less dissimilar in physical characters and chemical composition (see Table VI.), so that the manufacturer, by selection or mixture, can produce a variety of results. The section is also interesting as showing the frequent change in character of the material deposited at this place.

The black clay represents the most common type found in the Woodbine formation. The others, according to Hill, are exceptional and confined to the region around Denton.

More important, however, are the clays of the Tertiary beds, which occur near the base of the Lignitic stage, associated with the lignite beds, as well as higher up in the series, and even in the Marine stage. Their distribution is, in fact, practically coextensive with the deposits described under stoneware and fireclays, many of which can be, and some of which are already (as at Malakoff), used in the manufacture of pressed brick.

Those occurring interbedded with the lignites are usually shaly, often slightly carbonaceous, and have been more or less worked in the vicinity of Rockdale, Milam county. The value lies in the fact that, on account of the low iron-content, these clays burn at a comparatively low temperature to an agreeable buff color, as well as to a hard body. The depth below the surface is variable; at a few points the clays have been worked at the outcrop. The importance, however, will always be more or less dependent on the development of the lignite, since underground workings for the sake of the clay alone would not pay.

The other clays included in this group are those from Calaveras, Adkins and Rusk,—all occurring as lenticular deposits,

TABLE VI.—*Physical Tests and Chemical Composition*

Locality.	Laboratory Number.	Physical Tests.						
		Tensile Strength (Average).	Air-Shrinkage.	Cone 1. (1150° C.)		Cone 5. (1230° C.)		Fusion-Cone Number.
				Fire-Shrinkage.	Absorption.	Fire-Shrinkage.	Absorption	
		Lb. per Sq. In.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
Cisco, Eastland Co.......	944	[illegible]	[illegible]	[illegible]	[illegible]	5.4	0.421	...
Undershale, Minera, Webb Co.........	802	167	10.1	.....	......	Vitrified.	......	14
Undershale, Cannel, Webb Co..........	804	301	8.0	5.0	6.49	5.6	5.84	-27
Denton, Denton Co	842	173	8.6	5.0	0.7	7.0	0.5	...
Denton, Denton Co...............	843*a*	329	7.5	6.7	0.6	7.4	0.6	...
Denton, Denton Co............................	844	318	8 0	5.7	0 1	7.0	0.5	...
Calaveras, Wilson Co.........................	810	191	6.9	6.7	5.49	5.0	0.76	27
Adkins, Bexar Co...............................	815	161	7.6	0 6	13.78	1.4	11.17	14
Rockdale, Milam Co..........................	814	291	6.7	1.7	8 41	2.0	6.01	27
Rockdale, Milam Co..........................	827	189	11.6	.....	. ....	... .	......	28
Rockdale, Milam Co......	829	302	9.1	4.3	5.7	4.0	4.65	27
Rusk, Cherokee Co......................... .	856	261	10.3	2.3	9.35	6.3	5.0	27

a The analysis shows 16.6 per cent. of Fe_2O_3, which, if uniformly distributed, large grains, and the clay therefore

associated with sands, and on this account the point of mining operations would have to be shifted from time to time, if the demand for the clay were large. All three occurrences are somewhat semi-refractory, and the last two clays are quite siliceous—in fact, contain sand-streaks. The properties of these clays are given in Table VI.

(*b*) *Red- and Brown-Burning Brick-Clays.*—A considerable number of clay-deposits, scattered through the eastern half of Texas, are utilized for the manufacture of common or pressed red brick. These deposits probably represent a larger number of formations than those worked for any other kinds of clay-products in the State. The Carboniferous beds of northern Texas carry a number of beds of shale associated with the coal, and sometimes outcropping on the hill-slopes. These are worked at Thurber, Erath county; yield an excellent red brick for structural and paving purposes; and are also adapted to the manufacture of sewer-pipe. A section from the mine of the

of Buff-Burning Non-Calcareous Brick-Clays, Texas.

Color After Burning.	Chemical Composition.										
	SiO_2.	Al_2O_3	Fe_2O_3	CaO.	MgO.	K_2O.	Na_2O.	TiO_2.	H_2O.	Total.	Total Fluxes.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Buff..............	62.26	23.78	3.02	tr.	0.10	1.16	1.59	1.40	7 12	100.43	5.87
Buff	63.00	23.00	2.5	tr.	*1.1	1.7	0.3	1.3	4.8	97.7	5.6
Buff..	57.00	22.2	2.1	0 25	tr.	tr.	0.21	1.31	6.65	99.72	2.56
Buff....	57.00	25.59	3.44	0.96	0.72	0.94	0.82	0.87	10.00	100.34	6.98
Brown buff......	51.5	17 6	16.6	1.00	1.1	1.5	tr.	1.6	7.7	98.60	20.2
Buff........	56.2	23.7	1.5	0 6	1.5	1.4	2.2	1.6	11.1	99.8	7.2
Buff.........................	70.5	18.3	1.8	tr.	0.9	tr.	0.2	1.2	5 5	98.4	2.9
Buff..........................	68.7	15.9	3.3	3.1	0.5	tr.	0.3	1.4	5 9	99.1	7.2
Buff..........................	77 00	15 87	1.26	1.10	0.37	......	......	0.87	4.5	100.97	2.73
Buff..........................	64.00	22.59	1.22	0.88	1.15	tr.	0.06	1.51	5.80	97.21	3.31
Buff..........................	67.00	19.68	0.72	0 62	1.06	tr.	1.18	1.82	6.07	97.15	2.58
Buff..............	82.45	10.92	1.02	0.22	0.96	......	......	1.00	2.47	99 10	2.26

would indicate a deep-red burning clay. The iron, however, is segregated in burns to a buff body with red spots.

Bridgeport Coal Co. at Bridgeport, Wise county, illustrating the great abundance of shaly material in the Carboniferous beds, and its association with other kinds of sediments, shows: yellow sandy soil, 2 ft.; purple shaly clay, 3 ft.; whitish sandstone, 1 ft.; purple shaly clay, 10 ft.; blue shale, 70 ft.; limestone, 3 ft.; blue shale, 20 ft.; coal, 18 ft.; and bluish shale, 4 ft.

These Carboniferous shales probably represent one of the most important clay-resources of Texas; but they have not yet been thoroughly studied.

Next to the Carboniferous, the Upper Cretaceous contains a large quantity of red- or brown-burning clay, found in marls. It is probable that the Woodbine formation carries some clays of this character; but they are not worked.

The clays of the Eagle Ford, Taylor and Navarro formations resemble each other in some respects, being tough, dense, exceedingly plastic clays of low quality. Their peculiar charac-

TABLE VII.—*Physical Tests and Chemical Compo-*

Locality.	Geological Age.	Laboratory Number.	Physical Tests.							
			Tensile Strength (Average).	Air-Shrinkage.	Cone 05. (1050° C.)		Cone 1. (1150° C.)		Fusion-Cone Number	Color After Burning.
					Fire-Shrinkage.	Absorption.	Fire-Shrinkage.	Absorption.		
			Lb. per Sq. In.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.		
Thurber, Erath Co.	Carboniferous.	846	181	6.4	2.3	6 47	4	4.74	...	Brown.
Bridgeport, Wise Co.		930	196	7.3	7.7	0.15	......	......	2	Brown.
San Antonio, Bexar Co.	Upper Cretaceous.	811	169	9.3	......	......	......	......	5	Red-brown.
Austin, Travis Co.		876	.....	......	......	......	......	......	...	Red-brown.
Waco, McLennan Co.		946	182	5.3	0.7	15.82	5	2.81	3	Red-brown.
West Dallas, Dallas Co.		836	.....	This group of clays has an average air-shrinkage of 10.5%. The tensile strength ranged from 300–400 lbs. per sq. in. They all burn red-brown and fuse at cone 5-6. They have to be molded dry press.				......	...	......
8 m. W. of Dallas, Dallas Co.		841	.....					......	...	....
Sherman, Grayson Co.		875	.....					......	...	....
Paris, Lamar Co.		915	...					... ..	...	....
Cooper, Delta Co.		921	.....					......	5–6	....
Ferris, Ellis Co.		839	.....					......	...	....
Corsicana, Navarro Co.		835	487	12.4	5.0	6 10	5.5	5.43	5	Red-brown.
Greenville, Hunt Co.		863	74	6.0	0.4	20.30	1.0	16.60	12+	Brown.
Vogel Mine, Rockdale, Milam Co.		830	304	9 3	1.0	12.09	2.7	7.87	12	Red.
Elgin, Bastrop Co.	Tertiary.	958	355	8.0	1.6	7.75	3 0	3.61	12	Brown-buff.
Sulphur Springs, Hopkins Co.		869	315	9.3	1.4	14.96	5.4	5.46	5	Red-brown.
New Boston, Bowie Co.		860	192	11.6	6.3	2.73	12.20	5.77	5	Brown.
Houston, Harris Co.		826	316	9.3	0.4	6.63	0.8	5.43	5	Brown.
Houston, Harris Co.	Pleistocene.	824	159	10.1	0.7	15.29	4 0	8.82	3	Brown-buff.
Beaumont, Jefferson Co.		884	303	9.4	0.3	10.53	0.3	9.39	9	Brown.

ter makes them the most difficult clays in the State to handle, since their great toughness interferes with their being molded by any wet method, and therefore they have to be pulverized and pressed in a dry or nearly dry condition. This is necessary also because it gives the brick a more open structure, and permits the burning-out of the carbonaceous matter in the clay. Most of these clays show a medium to high lime-content, which is, however, irregularly distributed, and therefore does not exert any important influence on the color of the brick. On account of the size of the clay-deposits in these two formations, they can be worked more economically than any

sition of Red- or Brown-Burning Brick-Clays, Texas.

Chemical Composition.

SiO_2.	Al_2O_3.	Fe_2O_3	CaO.	MgO.	K_2O.	Na_2O.	TiO_2.	H_2O.	Miscellaneous.	Total.	Total Fluxes.
Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.		Per Cent.	Per Cent
68.75	15.81	4.05	0.60	1.64	tr.	0.08	0.60	4.07	Org., 2.10	97.70	6.
59.20	20.60	6.90	1.08	1 69	1.6	1.84	1.5	4.66	Org., 0.2	99.20	13.
59.47	18.24	4.77	4.3	tr.	tr.	0.24	1.14	5.70	CO_2, 3.25; SO_3, 0.90; Org., 0.55	98.56	9.
54.5	22.6	6.20	0.54	1.15	1.4	1.55	1.22	6.2	Org., 4.5	99.86	10.
72 36	7.84	1.72	6 48	2.23	1.2	1.7	0.12	3.72	CO_2, 3.30	100.67	13.
58.70	21.73	5.49	1.06	1.24	0.32	0.18	1.13	5.88	Org., 3.0; SO_3, 0.33	99.06	8.
53.21	22.83	6.48	1.70	1.72	tr.	0.05	1.75	7.06	Org., 2.0; CO_2, 3.10	....	9.
59.34	15.71	5.76	3.00	2.09	0.56	1.44	1.83	7.02	CO_2, 1.07; SO_3, 0.31; Org., 2.00	100.13	12.
64.2	20.13	1.87	0.34	1.62	0.69	1.78	2.00	5.32	Org., 0.1; SO_3, 1.8	....	......
53.48	17.84	3 16	8.08	1.44	0.85	1.6	1.0	6.9	CO_2, 4.66	99.01	15.1
49.60	16.06	5.60	10.66	2.14	1.38	0.77	0.7	5.02	SO_3, 1.12; CO_2, 6.94	99.39	19.
55.28	21.27	8.37	3 90	0.28	......	tr.	1.05	4.26	CO_2, 3.30; Org., 1.43	99.14	12.5
79.00	11.38	2.44	0.50	0.20	0.35	0.65	0.78	3.80	....	99.10	4.1
72.9	14.7	4.5	0.6	0.3	1.5	0.7	1.0	4.2	....	100.4	7.
70.4	17.3	1.8	1.0	tr.	0.6	2.2	0.8	5.4	....	99.5	5.
69.36	14.67	4.46	0.28	1.74	1.55	2.09	1.13	3.64	Org., 0.96; SO_3, tr.	99.88	10.1
66.01	18.82	6.33	0.55	1.88	0.16	0.08	0.95	4.8	....	99.58	9.
72.45	11.72	3.38	3.66	1.34	tr.	0.19	0.87	3.44	CO_2, undetermined.	97.05	8.5
49.40	17.90	4.50	9.50	1.88	......	tr.	1.05	4.58	CO_2, 9.55	98.36	15.
77.75	11.04	3.19	0.84	0.38	......	......	1.23	3.24	SO_3, 0.51	98.18	4.4

others in the State. Fig. 3 shows the pit of a brick-works at Ferris, Ellis county, at which the Taylor marl is exploited.

At most localities the clay is weathered to the depth of from 10 to 15 ft. This mellowed material is far easier to treat, but, owing to its limited quantity, it is commonly mixed in mining with the underlying beds of unweathered clay.

The Eagle Ford clays are worked at Paris, Sherman and Dallas, and the Taylor-Navarro marls at New Boston, Cooper, Greenville and Corsicana. Those at Greenville are more siliceous, and the clay, having a more open body, is easier to mold and burn than that of the other localities named. Several analyses of these clays are shown in Table VII.

Cretaceous clays are also worked about 3.5 miles east of San Antonio, but I am unable to say to which division of the Cretaceous these clays belong. In general character resemblance is shown to the Taylor-Navarro marls; except that numerous scattered bunches of selenite are present. So far as known, no deposits of clay are worked in the beds of the Will's Point stage.

The Lignitic clays, however, may prove valuable, and are already worked at Rockdale, where, overlying the lignites, red-burning, plastic, although somewhat siliceous, clays occur, which yield a red brick of excellent quality. At this place mining can be done from a surface-pit. Again, at Sulphur Springs, Hopkins county, and New Boston, Bowie county, there are red-burning shaly clays which may belong in the Lignitic stage, since the occurrence is well within its boundary, given on published maps. These clays differ much from the clay at Rockdale, being more shaly, bluish in color, and containing many concretions of limonite. Superficially some resemblance to the Navarro marl is shown; but the clays are less plastic, less calcareous, and more easily worked; possibly representing Upper Cretaceous islands, projecting through the Tertiary. At points intermediate between Sulphur Springs and Rockdale, either stoneware- or fire-clays, and no red-burning Lignitic clays are worked. Since many railroads cross the Lignitic belt, it would be desirable to prospect further for clays of this class, since those developed hitherto are more valuable than the Eagle Ford or Navarro-Taylor marls.

No red- or brown-burning brick-clays are worked in the other Tertiary areas, southeast of the Lignitic. Doubtless such exist, but the relative remoteness of the region from the main lines of transportation, and from the larger cities of the State, does not encourage development.

Of the Pleistocene clays there are many deposits, mostly rather siliceous. The better ones are to be sought in the area of the Beaumont clays, described on an earlier page. These seem, however, to be so variable in character that it is difficult to generalize the properties. All these clays are red- or brown-burning; but in chemical composition, shrinkage, fusibility, and other physical qualities, much variation is shown. Moreover, objectionable inclusions of lime-pebbles

frequently occur. Nevertheless, these clays are the only important source of brick-clay for the cities of Houston, Beaumont, and others in that region; and, if care be used in selection, a good product can be obtained. Analyses Nos. 824 and 826 in Table VII., both of clay from Houston, afford an excellent example of the difference in chemical composition of clay from adjoining yards.

(*c*) *Calcareous Brick-Clays.*—This class includes clays from different parts of the State, nearly all of which are used for brick. Excepting a Cretaceous clay southeast of Austin, all of those examined are Pleistocene alluvial deposits, underlying the terraces and flats along the rivers, especially the larger ones, such as the Colorado, Rio Grande, Brazos, etc. The two following sections, (1) from the river terrace at Austin, Travis county (Fig. 9), and (2) from Wharton, Wharton county, may illustrate their occurrence:

1. From the river-terrace at Austin: surface-soil, 1 ft.; silty clay, 7 ft.; and red, plastic clay, 16 ft., with gravel and sand below.

2. From Wharton, Wharton county: soil, 1 ft.; yellow clay, 3.5 ft.; chocolate clay, 1.5 ft.; yellowish clay, 15 ft.

Such sections, however, have no permanent value, since they change from day to day as the excavation uncovers different parts of the deposit. This variation is the result of the conditions of deposition. In order to preserve a general average quality of product, the "run of the bank"—*i.e.*, the product of a section through all layers—is usually shipped. At some localities the terrace-level is not above high water, and during periods of flood a fresh layer of silt, sometimes several inches in thickness, is deposited. This recent deposit may be quite different from the earlier ones, as at Gonzales, where the terrace-silts worked for brick are calcareous, while the present sediment of the river is ferruginous.

The calcareous silts are widely distributed throughout eastern Texas, and, on account of the ease of excavation and manipulation, this class of material is much used. The peculiar physical and chemical characters warrant the placing of the calcareous clays in a group by themselves. Summarizing the physical characteristics, we may say that but little water is absorbed and the plasticity is usually low; but the tensile strength

TABLE VIII.—*Physical Tests and Chemical*

Locality.	Laboratory Number.	Physical Tests.						
		Tensile Strength. (Average.)	Air-Shrinkage.	Cone 1. (1150° C.)		Cone 5. (1230° C.)		Fusion-Cone Number.
				Fire-Shrinkage.	Absorption.	Fire-Shrinkage.	Absorption.	
		Lb. Per Sq. In.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
Laredo, Webb Co	806	149	4.7	-1.0	19.34	4.3	8.3	7
D'Hanis, Medina Co	910	259	8.3	-0.4	19.63	......	....	1
Calaveras, Wilson Co	809	366	6.7	0	....	1.3	....	6
Seguin, Guadelupe Co	813	301	6.0	-1.3	....	-1.0	....	...
Gonzales, Gonzales Co	816	357	8.3	1.0	23.11	1.3	22.15	7
Gonzales, Gonzales Co	817	201	6.5	0.7	26.49	0	25.53	7
Austin, Travis Co	801	253	6 2	-0.7	... . .	1	23.49	7
3½ m. E. of Austin, Travis Co	945	225	10.6	7.3	12 3	.... .	. ... ..	5-6
Wharton, Wharton Co. Different layers.	901	119	0	-0.6	21.63	0	16.51	9
	902	330	10	0	12.01	......	.. ..	1
	903	308	6.6	-0.7	16.77	......	....	1
Taylor, Williamson Co	911	153	4.5	-1.0	54.5	-1	48.59	...
Brenham, Washington Co	873	355	12.8	. ...	... ...	......	. ...	3
Belton, Bell Co	918	340	6 6	0	21 92	0	21.1	...
Waco, McLennan Co	947	205	6	-0.3	15.01	7.4	3.5	2

is remarkably high. Of 17 samples tested, 8 showed an average of more than 300 lb., and 5 others exceeded 200 lb., per sq. in. Many exhibited high air-shrinkage, but all showed low fire-shrinkage and high porosity, until close to the vitrifying-point, when the body condensed suddenly. In the chemical analyses, the low silica-content, the high lime, and the usually high total fluxes, are specially noticeable. The percentage of lime is often astonishingly high; that of magnesia, on the other hand, is rarely so; and of the alkalies, soda averages higher than potash. Titanic acid is usually present, but rarely in excess of 1 per cent.

Interesting comparisons are afforded by an inspection of the totals of fluxes and the fusion-points. There does not appear to be even an approximate relation between the two. This is, no doubt, due to differences in texture, a coarse grain counterbalancing a high percentage of fluxes.

Composition of Calcareous Brick-Clays, Texas.

Color After Burning.	Chemical Composition.											
	SiO_2.	Al_2O_3.	Fe_2O_3.	CaO.	MgO.	K_2O.	Na_2O.	TiO_2.	H_2O.	CO_2.	Total.	Total Fluxes.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Buff..........	59.03	11.19	2.77	12 16	0.8	tr.	0.18	1.05	2.1	9.6	98.88	15.91
Brown buff......	51 12	11.04	4.1	14 24	0.9	0.4	1.59	0.96	4.0	10.62	98.97	21.23
Buff.	37.45	7.72	2.02	27.92	0.36	......	......	tr.	2.4	21.8	99.67	30.3
Brown buff.....	18.62	3.23	1.26	41.30	0.42	......	......	tr.	2.42	32.5	99.75	42.98
Buff..................	37.05	8.13	1.8	25.3	2.23	tr.	0.08	0.47	2.64	22.12	99.82	29.41
Brown buff......	41.2	6.5	1.98	24.4	1.62	0.09	0.14	0.43	2.28	20.63	99.27	28.23
Brown buff.....	53.6	9.0	2.6	17.8	1.2	1.8	tr.	0.8	2.72	11.64	101.16	23.4
Buff........	34.6	15.02	3.02	21.48	0.15	1.43	1.34	0.96	6.0	15 6	99.6	27.42
Brown buff......	67.03	10.57	1.94	9.33	1.23	......	......	0.90	2.65	7.66	101.31	12.5
Brown buff......	63.56	8.18	4.32	10.0	0.15	tr.	1 0	0.95	4.16	7.36	99 68	15.47
Brown buff......	65.07	9.16	2.8	8.44	0.21	0.5	1.66	1.05	3.72	6 92	99.53	13.61
Buff..................	21.72	7.97	2.23	36.54	0.95	tr.	tr.	0.52	2.06	28 44	100 43	39.72
Brown buff......	51.3	14.4	6 2	10.3	tr.	tr.	4 1	0.8	4.9	7.6	99.6	20.6
Buff.	47.2	4.1	2.4	21.0	1.4	tr.	1.3	0.7	2.9	18.1	99.1	26.1
Brown buff......	71.4	8.2	2.3	6.34	2.44	1.22	1.6	0.14	3.25	3.7	100.59	14.9

As is well known, calcareous clays are very porous after burning at low temperatures and before fire-shrinkage begins. This degree of porosity appears to be in direct relation, not to the amount of CO_2 present, but rather to the combined percentages of CO_2 and H_2O.

The calcareous clays are much worked in Texas for making common brick, pressed brick, and some drain-tile. Laredo and Austin are the most important localities.

(*d*) *Sandy Brick-Clays.*—These represent the poorest quality of material used for the manufacture of clay-products in Texas. The clays are sandy, or rather clayey sands, usually of Pleistocene age, although possibly a few may belong to the Tertiary. In some instances, as at Longview and Harrisburg, the clay is interstratified or underlain by a bed of more plastic clay, which is mined with it. It might be sufficient to pass these materials by, with no further statement than to refer to the use for com-

TABLE IX.—*Physical and Chemica*

Locality.	Laboratory Number.	Physical Tests.						
		Tensile Strength. (Average).	Air-Shrinkage.	Cone 05. (1050° C.) Fire-Shrinkage.	Cone 05. (1050° C.) Absorption.	Cone 1. (1150° C) Fire-Shrinkage.	Cone 1. (1150° C) Absorption.	Fusion-Cone Number.
		Lb. Per Sq. In.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
Elgin, Bastrop Co.	959	455	9.6	0.4	9.53	2.3	5.78	9
Fulshear, Fort Bend Co	937	133	5.6	0	12.82	0.3	12.29	...
Houston, Harris Co.	825	260	6.2	-0.6	10.12	-0.6	9.44	12+
Top and middle bed, Harrisburg, Harris Co.	821	188	4.8	-0.3	14.83	-0.3	14.38	...
Lower bed, Harrisburg, Harris Co.	823	275	8.6	0.3	9 98	0.3	9.95	9
Cedar Bayou, Chambers Co.	866	200	6.6	-0.7	12.04	0	12.56	...
Colmesneil, Tyler Co.	904	77	4 0	-0.3	9.14	0	9.45	14+
Giddings, Lee Co.	909	234	4.3	-0.4	11.20	-0.3	11.03	9
Cleburne, Johnson Co.	922	128	4.0	0	13.58	-0.7	13 63	...
Rusk, Cherokee Co.	857	207	9.6	0.4	13.41	1.7	10.82	14
Winnsboro, Wood Co.	865	315	6.0	0	10.69	0	10 62	14+
Top and middle layer, Longview, Gregg Co.	905	177	10.3	0.6	15 01	3.7	10.45	14-15
Lower layer, Longview, Gregg Co.	906	206	10.5	0.7	14.63	3	8.80	14
Detroit, Red River Co.	920	262	10.3	0	9.73	0.7	9.93	9
Marshall, Harrison Co.	907	122	3.3	-0.3	13.74	0	12.96	14+
Texarkana, Bowie Co.	859	117	6.6	0	11.77	0	13.27	14+

mon brick; but the tabulated summary of the tests in Table IX. is not without interest. The high percentage of silica is, perhaps, the most prominent feature of the composition; and it seems remarkable that a material carrying from 85 to 90 per cent. of silica should possess sufficient plasticity to permit of its being molded into bricks.

The alumina is variable. If we assumed most of it to be present as kaolinite, it would indicate that at least 70 or 80 per cent. of the clay is sand and the balance clay. The fluxing materials are all low (having been mostly leached out, if they were ever present), and what iron oxide there is forms a coating around the quartz-grains. Titanium is rather high. In spite of the siliceous character, some of these clays (as, for example, those from Elgin and Winnsboro) show a remarkably high tensile strength when air-dried, and all show the low

Analyses of Sandy Brick-Clays, Texas.

Color After Burning.	Chemical Composition.										
	SiO_2	Al_2O_3.	Fe_2O_3.	CaO.	MgO.	K_2O.	Na_2O.	TiO_2.	H_2O.	Total.	Total Fluxes.
	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Red-brown	72.7	9.5	4.1	4.1	0 8	2.4	tr.	0.6	4.5	99.1	11.4
Brown..................	83.80	9.23	2.3	tr.	tr.	0 56	0.54	0.87	3.1	100.4	3.4
Red......................	89.00	3 69	1.65	0.47	0.65	tr.	0 06	0 84	1.62	97.98	2.83
Red......................	80.39	9.82	2.88	0.42	0.45	tr.	0.19	0.35	3.11	97.61	3.94
Red......................	80.84	8.09	2.25	1.44	0.26	tr.	0.10	0.78	6.0	99.76	5.05
Red-brown	85.60	6.71	1.44	tr.	0.43	0 5	0.65	1.0	3.1	99.43	3.02
Brown..................	90.00	4.6	1.44	0.1	0 1	tr.	tr.	0.7	3.04	99.98	1.64
Brown..................	81.50	5.43	3.6	1.3	0.25	0.49	1.56	0.87	4.0	99.	7.2
Brown..................	87.3	4.06	3.52	1.03	0.10	tr.	0.2	0.48	2.44	99.13	4.85
Brown	72.76	14.46	3.81	0.08	1.93	tr.	tr.	1.43	4.61	99.08	5.82
Brown..................	85.35	6.72	1.87	0.4	0.24	tr.	0.2	1.01	3.2	98.99	2.71
Brown..................	73.06	9.88	6.92	1.5	0.25	tr.	0.12	1.0	6.64	99.37	8.81
Brown	68.5	18.41	3.02	0.7	1.05	0.47	0 91	1.31	6.2	100.57	6.15
Brown..................	78 5	10.5	3.6	0.45	0.23	0.9	0 4	0.32	4.22	99.12	5.58
Red-brown	83.9	5.52	4.75	0.4	1.32	0.15	0.45	1.57	2.44	100.5	7.07
Red......................	88.71	4.88	2.0	0.3	0.97	tr.	tr.	0.9	2.28	100.04	3.27

fire-shrinkage characteristic of sandy clays, some even swelling slightly at the lower temperatures, because of the high silica-content. A careful comparison of the chemical analyses and physical properties will, as usual, show that the former give us but little information regarding the latter. Table IX. gives the physical and chemical properties of a number of samples examined.

(*e*) *Paving-Brick Clays.*—Only one factory in Texas now produces paving-bricks, most of the supply being brought from other States. This has naturally raised the question whether suitable clays are scarce in Texas. Few have been found hitherto, and all of these are in the Carboniferous, no clays of vitrifying quality having been observed in the Cretaceous, Tertiary or Pleistocene, east of the 99th Meridian. Those now known come from Graham, Young county, and Thurber, Erath

TABLE X.—*Physical Tests and Chemical*

Locality	Laboratory Number.	Physical Tests.						
		Tensile Strength. (Average.)	Air-Shrinkage.	Cone 05. (1050° C.)		Cone 1. (1150° C.)		Fusion-Cone Number.
				Fire-Shrinkage.	Absorption.	Fire-Shrinkage.	Absorption.	
		Lb. Per Sq. In.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
M. K. Graham property, Graham, Young Co.....	880	197.5	6.1	7.7	4.5	8.6	0.90	9
W. D. & C. H. Craig property, 16 m. W. of Graham..........	879	200	7.3	3	8.35	6	1.54	9
14 m. W. of Graham, Young Co........	878	188.3	6.2	2.7	10.91	8	0.13	9
Thurber, Erath Co., No. 1 Hill.	847	299	8	5	4	5.3	0.14	3
Thurber, Erath Co., Valley Clay......	848	333	7.7	5.6	3.58	6.3	0.10	5

county, and agree in being of dense-burning character, low fusibility and high plasticity. Up to the present time only those at Thurber have been worked. Here two distinct beds are found, the one being near the summit of what is known as No. 1 Hill, and at least 75 ft. higher than the coal, while the other is found almost immediately below the surface over a large area in the valley to the north of the brick-works.

Table X. shows the tests and analyses of the paving-brick clays.

4. *Slip-Clays.*

At several localities beds of clay are found, which, on account of the high percentage of fluxing-impurities, especially alkalies, fuse at a comparatively low temperature, and run to a colored glass or natural glaze. Clays of this type are used for glazing stoneware, and their discovery is always of importance, since the main source of supply of American slip-clay is New York State, whence the material is shipped to many parts of the country. All of the slip-clays found during the field-work of the survey belong to the Pleistocene, with one exception, which is Carboniferous.

The Texas slip-clays occur in beds of variable thickness, interstratified with sands or other clays, and there is nothing in the appearance to indicate the easily fusible character. It

Composition of Paving-Brick Clays, Texas.

	Chemical Composition.											
Color After Burning.	SiO_2.	Al_2O_3.	Fe_2O_3.	CaO.	MgO.	K_2O.	Na_2O.	TiO_2.	H_2O.	Moist	Total	Total Fluxes.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Buff..................	63.3	24 5	1.3	0.1	0.2	2.3	1 3	1.0	6.1	2.6	100.1	5.2
Light brown...	......	.. ...	...	......	......	Not	analy	zed.	... ..	......	......	......
Brown..............	60.5	19 4	9.3	tr.	0.3	1.4	1.5	0.6	6.5	2 8	99 5	12.5
Brown..............	64.52	17.72	4 46	0.27	1.58	2.71	1.24	1 30	5 44	. . . .	99.24	10.26
Red-brown......	63.07	19.43	4.75	1.32	0.50	......	......	1.47	6.90	0.19	99.09	6.57

is therefore only by accident or detailed investigation that the special qualities have been recognized; and, in a few instances, have become known to potters.

Such clays have thus far been found near Carmona, Polk county; Navasota, Grimes county; San Antonio, Bexar county; and Bridgeport, Wise county. That near San Antonio is perhaps the best known, and is found outcropping in the banks of Alazan creek, the section showing: black soil, 2.5 ft.; impure, yellow, gravelly clay, 2.5 ft.; and yellowish-brown slip-clay, 7.5 ft., with sand below.

The same type of clay outcrops again on the Leon creek, 7 miles south of San Antonio.

The slip-clay northeast of Navasota is a white chalk-like material, occurring in beds from 6 to 8 ft. thick, with but little overburden; that found 2.5 miles southeast of Carmona is at least 6 ft. thick, with little overburden; thus, in every instance, the deposits are sufficiently thick to supply a large demand. Table XI. gives the analyses of the Texas slip-clays.

IX. The Texas Clay-Working Industry.

In 1904, the clay-products of Texas placed it the eighteenth in that department among the States of the Union, most of its product consisting of common building-brick, made chiefly for

TABLE XI.—*Analyses of Slip-Clays, Texas.*

	2 m. S. W. of Carmona, Polk Co.	Leon Creek, San Antonio, Bexar Co.	Alazan Creek, San Antonio, Bexar Co.	13 m. N. of Navasota, Grimes Co.	Bridgewater, Wise Co.
Laboratory No.	950	949	924	951	930
SiO_2	68.34	38.08	57.01	68.56	59.20
Al_2O_3	15.28	11.36	11.85	18.53	20.60
Fe_2O_3	3.44	2.6	3.02	0.72	6.90
CaO	1.20	23.70	9.56	0.60	1.08
MgO	0.88	tr.	1.20	0.12	1.62
K_2O	2.47	0.58	0.75	2.27	1.60
Na_2O	3.55	1.6	2.01	2.72	1.84
TiO_2	0.52	0.7	1.13	0.43	1.50
H_2O	4.7	3.06	4.00	7.00	4.66
CO_2	......	18.80	8.00	......	......
Total	100.38	100.44	98.53	100.95	99.00
Total fluxes	11.54	28.48	16.54	6.43	13.04

local use. Some pressed brick, paving-brick, stoneware, fire-brick and drain-tile were also made; but the quantity was small. Sewer-pipe, terra-cotta, fire-proofing, conduit- and floor-tiles were not made at all.

The prominence of a State as a manufacturer of clay-products, depends not only upon its suitable raw material, but also upon its available markets. Texas has the former in abundance, and as to the latter, it may be said that the demand is increasing, but is still largely supplied by establishments outside of the State. It remains, therefore, for enterprising parties to develop the clays, and not only get control of the local markets, but supply those of the other Gulf States and Mexico as well.

A New Colorimeter for the Determination of Carbon in Steel.

BY CHARLES H. WHITE, CAMBRIDGE, MASS.*

(London Meeting, July 1906.)

METHODS in colorimetry are based on the assumption that the intensity of the color of a definite volume of solution is directly proportional to the quantity of the color-producing substance present. In the preparation and examination of colored solutions there are three ways in which the depth of color may be varied. These variables are: (1) the quantity of the coloring matter; (2) the quantity of the solvent, or the dilution; and (3) the thickness or depth of that portion of the solution examined. With these three variables as a basis, three general methods in colorimetry have been devised, all of which have been used in the determination of carbon in steel.

In the first method, the quantity of the solution is kept the same for both the unknown steel and the standard, and the sections compared are of equal thickness, while the quantity of the standard is varied until the color obtained is of the same intensity as that produced by a definite quantity of the unknown steel. In laboratories where this method is used there is prepared and kept for permanent use a series of solutions with varying amounts of the standard, representing the percentages of carbon from the lowest to the highest demanded in that laboratory. These standard solutions are kept in bottles or test-tubes of equal diameter, and are arranged in series from the faintest to the most intense color, so that it is a simple matter to find the place of the unknown steel in the series, when the solution is of a definite volume and is contained in a tube of the same diameter as the standard tubes.

In the second method, equal quantities of the standard and of the unknown steel are dissolved, the standard is diluted to a definite volume, and the solution of the unknown steel is di-

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luted until the colors are of equal intensity in sections of equal thickness. This method is usually carried out by dissolving equal quantities of the two steels in equal volumes of dilute nitric acid, and then diluting in long graduated test-tubes of equal diameter until the colors agree when the light passes through the tubes at right angles to their length. The percentage of carbon in each is then proportional to its volume.

In the third method, equal quantities of the two steels are dissolved in equal volumes of the solvent, and the thickness or depth of the solutions under examination is varied until the colors are the same. The percentage of carbon is then inversely proportional to the thickness of the section. This method has been applied to the determination of carbon in steel by pouring varying amounts of the two solutions into long graduated test-tubes with flat bottoms, until the colors are of equal intensity when viewed longitudinally from above, the source of light being below the tubes.

Each of these methods has excellent points to recommend it; but, as is well known to experimenters in colorimetry, there are serious objections to all of them, as they have hitherto been applied in practice. In the application of the first method, it seems almost impossible to obtain permanent standards. It is not practicable to keep on hand standards for a wide range of steels; and the standard for one kind of steel will not serve as a standard for a steel of a different kind, or for one that has been subjected to a different treatment. The strongest objection to the second method is the necessity for repeated dilutions by guess, mixing and comparing until the correct volumes are more or less accidentally hit upon. Moreover, if the graduation of the tubes is not marked in such a manner as to correspond in numerical value to the hundredths per cent. of carbon present, there is a loss of time in arithmetical calculations. For the third method, the difficulty of obtaining satisfactory apparatus has been serious. A moderately long tube having a flat bottom with true plane surfaces at right-angles to the axis is made with great difficulty; and although the apparatus may be so constructed that the depth of the solution may be increased or diminished conveniently, the emptying and cleaning after each determination renders manipulation necessarily slow.

The colorimeter here described was designed for the rapid

and accurate application of the third method, and has proved itself free from many of the objections belonging to those previously devised. In this method, equal quantities of the standard and of the unknown steel are dissolved in equal volumes of the solvent, diluted to a definite volume, and varying thicknesses of the solutions compared until the colors agree; then the ratio of the thicknesses of the two solutions compared is inversely

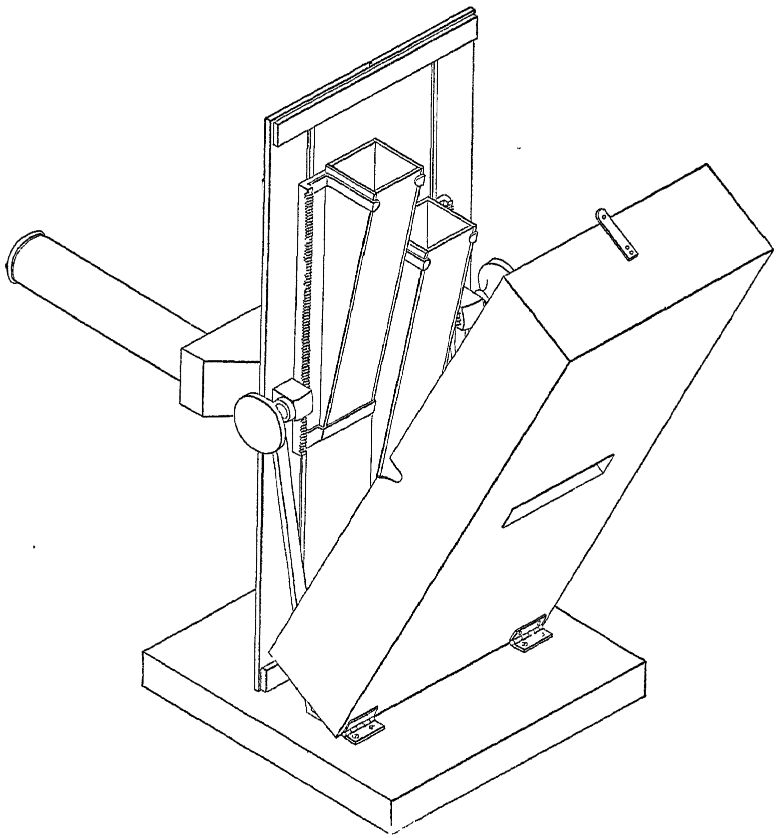

FIG. 1.—VIEW OF COLORIMETER OPENED TO SHOW WEDGES AND RACKS.

proportional to the percentages of carbon present. In using this apparatus, the two steels—the standard and the unknown—are dissolved in the usual manner in large test-tubes, diluted to equal volume, indicated by a mark on the tubes, and the solutions are compared in hollow glass wedges. The wedges must be of equal angle, and it is convenient to have them as nearly as possible of equal size. The large end is left open for the in-

troduction of the solutions, and they are mounted side by side in a vertical position for comparison. Each wedge may be raised or lowered by means of a thumb-screw, carrying a pinion which engages a rack holding the wedge. (See Figs. 1 and 2.) The wedges are incased in a box, which is provided with a narrow horizontal aperture in both the front and the back,

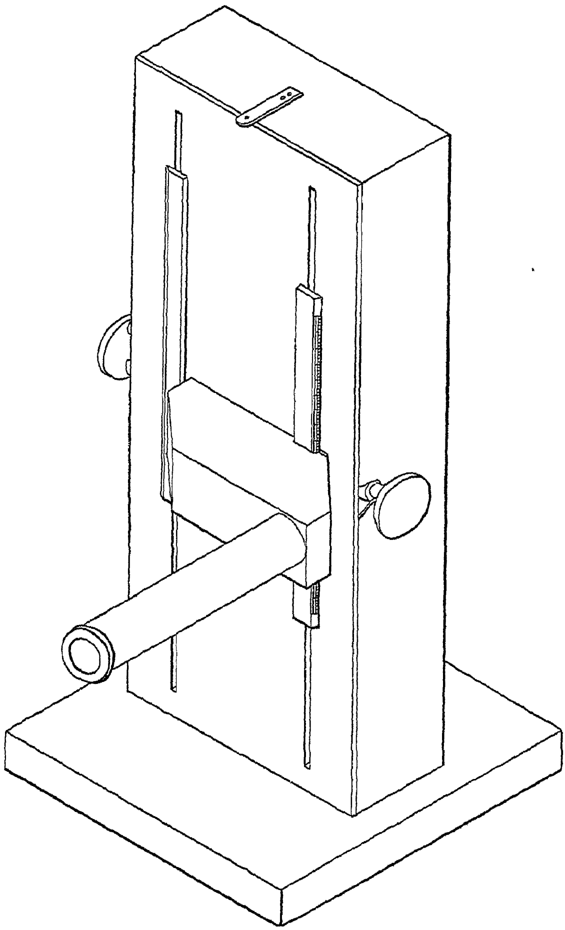

Fig. 2.—View of Colorimeter Closed.

admitting the direct passage of the light through the wedges in this zone only. In front of the aperture, or slit, are three mirrors, shown in Fig. 3, arranged in such a manner as to render the comparison of the colors easy and exact. The band of color, as observed in mirror *C* through the tube, *T*, is composed of three nearly-equal parts. The central portion is the

color of the solution in wedge *S*, reflected from mirror *A*, through that portion of mirror *B* from which the silver has been removed. The outer portions of the band of color are from wedge *U*, and are reflected from the silvered portions of mirror *B*. The racks which carry the wedges carry also graduated scales on the front of the instrument, which show, in every position, the distance from the sharp edge of the wedge to the slit where the comparison is being made. Uniform illumination is secured by admitting the light to the solutions through a pane of ground-glass, shown in section in Fig. 3.

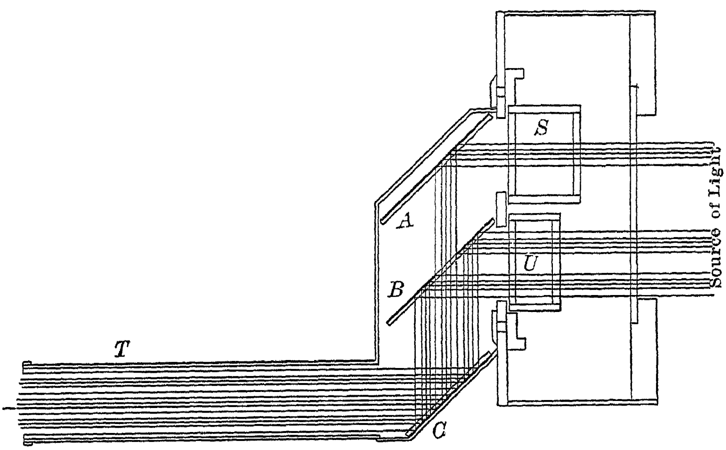

FIG. 3.—SECTIONAL PLAN OF COLORIMETER.

In operating this instrument, if the wedges are so adjusted that the colors are of equal intensity, and if their thickness at the slit—the point of comparison—is determined, or the ratio of one to the other is known, it is easy to calculate the percentage of carbon in the unknown steel. Since a longitudinal vertical section of one of these wedges is triangular and equal in all respects to a similar section of the other wedge, we are not restricted to the measure of the thickness of the solutions to determine the desired ratio, but the length of each from its sharp edge to the point compared may be read off on its graduated scale and used instead.

Suppose, for example, the standard steel has 0.30 per cent. of carbon, and, after the wedges have been adjusted by means of the thumb-screws until the colors match, it is found that the scale on the side of the standard reads 72 and the scale of the

other wedge reads 60; then the proportion $(60:72 = 0.30:x)$ shows that the percentage of carbon in the steel under examination is 0.36. But the solutions in the wedges may be as easily compared at the graduations 30 and 36 as at 60 and 72; and if the scale on the side of the undetermined steel is set at 30 and the other wedge, carrying the standard, is adjusted until the colors are equal, then its scale must read 36,—the desired figure.

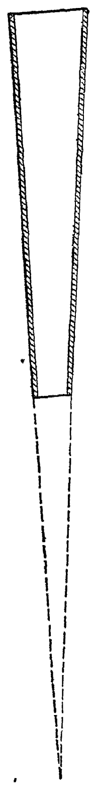

FIG. 4.—SHORT FORM OF WEDGE.

It is obvious that with this instrument we may vary the standard at will from the highest to the lowest in carbon, and also avoid the difficulties attending attempts at permanent standards; the haphazard diluting required in the second method is obviated, while the wedges are as easily emptied and filled as test-tubes. The chief advantages, however, derived from the use of this instrument are the speed and accuracy with which the colors are matched, and the fact that the percentage of carbon is read off directly.

With respect to the best form of wedge, it is evident that, other things being equal, the accuracy of the instrument increases as the angle of the wedge diminishes. But very thin sections of colored solutions are not easily matched, and long wedges are unwieldy and in many ways objectionable. It is not necessary, however, to use the whole length of the wedge; for the standard and the unknown steel cannot differ greatly in the amount of carbon contained, nor, therefore, in the colors produced when in solution. The wedges, then, may be designed to have any suitable angle, and only so much of the large end need be used as will afford the range desired. In other words, the wedges are cut off, the lower end of the upper portion sealed, and the lower part discarded, as shown in Fig. 4. The scales, however, must be graduated to accord with wedges of the full length. Wedges of this type are useful for the most exact colorimetric work, but with the ordinary size shown in the instrument in Fig. 1, comparisons are quickly made and read with accuracy to the second decimal place, which is as accurate as other parts of the color-method for carbon in steel can be relied upon to be.

A Device for Regulating the Discharge of Water from a Reservoir.

BY P. BOUÉRY, WEAVERVILLE, CAL.

(London Meeting, July, 1906.)

THIS account of a contrivance which has been found serviceable in practice may be of interest to engineers, and especially to those engaged in hydraulic mining.

In that process, one feature which seems at first sight to possess little significance, is highly important in its effects—namely, the regular supply of water to the "giants." The chief condition of this regularity is an even flow in the pipe-line from the penstock to the nozzle. This, of course, requires that the penstock be kept full. Any considerable surplus of water, overflowing at the penstock, is a simple waste, unless it be otherwise utilized in the sluices. At all events, it does not affect the delivery of the nozzles. But a deficiency at the penstock may admit air into the pipe; and this may produce hydraulic recoil, injuring the line by incessant vibrations, giving rise to leaks, and even to blow-outs, and (when a partial vacuum exists at the time of the fracture) the collapse of the pipe.

An accident of this sort in our main pipe-line, occasioned partly by a settling of the ground and partly by the presence of air in the pipe, led to the employment of the device here described. The line referred to is made of 30-in. pipe, successively reduced to 18 and 15 in. in diameter, for the supply of two giants. By reason of the causes above mentioned, the 30-in. pipe burst more than 1,400 ft. from the penstock, and, despite the air-valve, the 30-in. pipe was collapsed to such a degree that one length of it, next to the penstock, was squeezed together and positively swallowed by the main pipe, through which it traveled for 750 ft., piercing through a short elbow on the way.

This remarkable result was undoubtedly the effect of the vacuum suddenly produced when the pipe burst.

Aside from the dilatation and contraction due to changes in temperature, and changes in the ground itself (all of which, except earthquakes, should be prevented by suitable care in construction), a pipe-line, properly laid, is exposed to no other cause of injury than the action of air mixed with the water, which, by reason of the difference of the two in specific gravity, produces variable and unequal strains in the pipe.

At every hydraulic mine, a reservoir is interposed, if practicable, between the ditch and the pipe-line to the giants, for the storage and proper utilization of the water. When this reservoir is full, and the giants are ready to be operated, the gate-tender opens the main gate of the reservoir just enough to supply the giants and give a little overflow at the penstock, to prevent the entry of air. If the supply to the reservoir is smaller than the discharge thus permitted, the consequent lowering of the water-level in the reservoir reduces the hydrostatic pressure at the discharge-opening, and therefore the rate of discharge. To counteract this effect, the gate-tender raises the gate so as to compensate by greater area the diminished pressure; and this process is continued until the reservoir is almost empty.

This method of regulating the supply of water to the giants evidently leaves the whole matter to the vigilance and skill of one man, who, however faithful in the main, cannot be expected to prevent, or even to be able to prevent, irregularities in the working-supply. If he raises the gate either a little too high or a little too late, the result is in one case an unnecessary loss of precious water through excessive overflow, or, in the other case, the entrance of air into the pipe-line, with consequent injurious vibrations.

We have found that this source of irregularity can be removed, while, at the same time, the wages of one man for both the day- and the night-shift can be saved, by utilizing the water in the reservoir itself to furnish a hydraulic head sufficient to operate, automatically and continuously, a little auxiliary gate (independent of the main gate), and thus maintain the small overflow at the penstock, needed to prevent the entrance of air. The device adopted for this purpose is shown in Fig. 1. This sketch is not drawn to scale, but it will sufficiently illustrate the following description:

1. In the dam of the reservoir, we cut a second outlet, of appropriate size, at the level of the main gate.

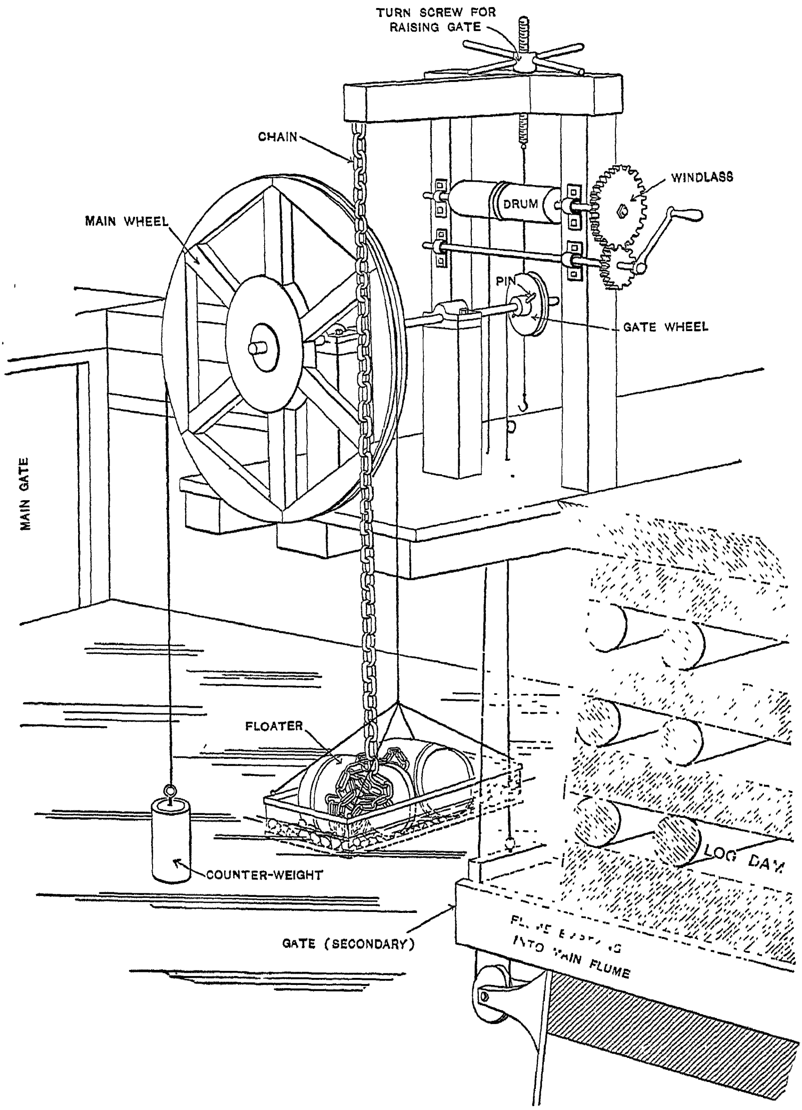

FIG. 1.—SKETCH OF AUXILIARY APPARATUS FOR REGULATING THE DISCHARGE OF WATER.

2. On the top of the dam we set up a large wheel, with grooves for two small cables, wound in opposition, and supporting respectively a floater and a counter-weight.

3. On the prolongation of the shaft of the main wheel we fixed a smaller "gate-wheel," of relative diameter duly calculated, carrying a cable, attached at its lower end to a secondary gate, opening a flume emptying into the main flume. This cable moves in a direction the reverse of that of the cable carried by the main wheel and which supports the floater. The small gate-wheel is loose on the shaft, but can be made, by means of a pin passing through hub and shaft, to revolve with the main wheel.

4. Under the small secondary gate we placed another wheel, around which passed a cable attached at one end to the bottom of the said gate, and at the other end to the drum of a windlass above.

5. Finally, above the small gate-wheel, we placed a turn-screw, by means of which the small secondary gate could be raised independently of the automatic operation of the apparatus. It is evident that, as the floater suspended from the main wheel falls with the water-level in the reservoir, the consequent revolution of the gate-wheel will raise the secondary gate, and thus increase the amount of water discharged at a lower head. The proper relation of this compensating increase is determined by the relative diameters of the two wheels, of which more will be said presently.

6. When the reservoir is full, the main gate is raised, and the small gate-wheel is pinned to the main shaft. If the water leaves the reservoir in excess of the supply, the consequent lowering of the water-surface will depress the floater, turning the main wheel, and with it the small gate-wheel. Both the cables concerned will be in high tension: one supporting the descending floater; the other raising the secondary gate; consequently, neither can, under ordinary circumstances, fail to operate. Moreover, there is no sudden change of the aperture of the secondary discharge, which is increased proportionally to the gradual descent of the floater, until the reservoir is empty. When this has taken place, the small gate-wheel is loosened by the removal of the pin; the secondary gate is closed by means of the windlass; and, the main gate being also closed, the reservoir is allowed to fill again. During this process, the cable supporting the floater is kept in tension by the counter-weight.

It is easy to calculate for each case the pressure on the small gate to be overcome, and the force required to overcome it in raising that gate; the ratio of diameters between the main wheel and the auxiliary gate-wheel, required to secure the necessary leverage; and the size and weight of floater required to overcome all friction and raise the small gate. In this operation, however, the strain is greatest at the beginning, and diminishes to a minimum at the end. This variation is counteracted, and the water-line is kept at about the same place during the raising of the gate, by means of a heavy chain, hung above the floater, and long enough to reach it when the water is very low. As the floater descends, the amount of chain carried by it decreases, to the extent required for a uniform action.

The calculation of the size of the auxiliary gate can be easily made when the reservoir is regular in shape. But for a reservoir of complicated form, the necessary data must be obtained by observation and measurement. For this purpose the rise of the main gate is noted every fifteen minutes after the start. The width of the gate and the amount of each rise are the sides of the rectangle for the opening. By combining these periodic observations, the total area of the main gate-opening is obtained; and the size and form of the small gate-opening can be planned with sufficient accuracy.

The cost of the material of this device does not exceed $100. In fact, we used an old Pelton wheel, enlarged in diameter to 8 ft., with a rim made of planks for the grooves. The floater was made with two empty barrels attached to a frame supporting a box full of boulders. Some old pieces of chain, welded together, with a pulley and a windlass as the only new material, completed the system.

The results obtained thus far have saved us the wages of one man (by enabling one tender to do the work formerly requiring two), and 50 per cent. of the cost of keeping up the pipe-line. Some better arrangement might be made, but it would probably be much more expensive, and this process may be of some value in remote countries where cheap installations are required. We are satisfied that the device could be made simpler in construction. Its installation was intended only as an experiment; but its operation has been so satisfactory that the temporary structure has become final.

The Cyanidation of Raw Pyritic Concentrates.

BY FRANK C. SMITH, WENDENDALE, ARIZ.

(London Meeting, July, 1906.)

THE following article covers the history of a metallurgical campaign, commenced in March, 1905, at the mines of the Socorro Gold Co., in the so-called desert region of Yuma county, Arizona. The results obtained were simply a function of necessity, as will be seen from the description of the local economic conditions and the character of the ores treated.

The mines and mill of the Socorro Gold Co. are located near the small village of Harrisburg, about 60 miles from Congress Junction, Ariz., which, at the time this work was undertaken, was the nearest railroad station, and from which all supplies were brought by wagon through the desert, at a cost varying from $10 to $15 per ton in addition to the railroad freights. Wood, the only available fuel, cost $7 per cord. Water for all purposes was pumped to the plant from a well in the valley about 4 miles away. Under the Arizona law, a working-shift is 8 hr. Wages were $6 per day for foremen and mill-men, $4 per day for blacksmiths and engineers, $3.50 per day for miners, and $3 per day for shovelers and trammers.

The ore consisted of quartz, in which, above the 250-ft. level, the iron-minerals were largely oxidized and some free gold was visible; below that level few traces of oxidation occurred, and pyrite constituted the principal mineralizer in the quartz, together with occasional pockets of galena and a few eccentric specks of covellite. The 20-stamp mill was equipped with plates for amalgamation, and three Standard tables for concentration. The tails from the tables were elevated by a centrifugal pump to a launder, by which they were conveyed to a 60-ton cyanide-mill for further treatment. The cyanide-mill was erected at so great a distance from the stamp-mill that a separate crew of workers and a separate power-plant were necessary. No arrangement had been made for the treatment of the concen-

trates as produced; although later, an adobe roasting-furnace had been erected near the stamp-mill, as well as a small cyanide-plant, for the treatment of the roasted product.

The treatment of the mill-tails in the remote cyanide-plant was unprofitable. The concentrates could not be shipped for treatment, on account of the high freight-cost and high smelter-charges; and, as the law of Arizona prohibited the use of wood as a fuel for roasting ore, the adobe furnace was useless. As a natural conclusion, it seemed evident that, if possible, an entire change of programme was necessary, involving two requisites: 1. A very clean concentration of the mill-tails, producing a final material containing very low values; 2. A local treatment of the raw concentrates.

The concentrating-plant was unfitted to attain the first requisite; the Standard tables affording a very fair rough concentration, but making a very poor separation of middlings and slimes. Two Frue vanners were installed, the inefficient middling-pumps from the tables were thrown on the scrap-heap, and the plant started operation as follows: The tails from 10 stamps passed from the plates to one Standard table, and from this upon a second table. The tails from the other 10 stamps were similarly conducted upon a Standard table, but from this into a small *spitzkasten*, situated above the distributor of Frue vanner No. 1, from which the thickened pulp passed to the vanner for concentration, while the overflow passed to the slime-sump, mentioned later. The slimes from each of the tables were conducted by launder into a sump, from which they were elevated by a 2-in. Byron Jackson centrifugal pump into a set of 3 large *spitzkasten* arranged in a series, the overflow from the first passing into the second, etc., the final overflow, consisting of fairly clean water, passing into a supply-tank for use as feed-water for the batteries. The more or less de-watered slimes from the large *spitzkasten* were conducted by launders to a small *spitzkasten*, which discharged a product containing still less water upon Frue vanner No. 2 for concentration, the overflow from this last *spitzkasten* also passing back to the slime-sump. In the case of the first 10 stamps, the middlings from the first table passed with the tails to the second table, from which the final middling-product was caught in a separate receptacle and returned to the battery. The middlings

from the second 10 stamps passed with the tails from the table to the vanner. By this arrangement, only that water was wasted which was necessary to carry off the final tails and that which was necessary for the vanner-supply. It was found that the final tails were of very low grade; the result being somewhat surprising, considering the smallness of the concentration-plant for an output of 70 tons per day, but being doubtless referable to the comparatively simple character of the ore.

A futile attempt had formerly been made to extract the values of the table-concentrates by cyanidation without roasting, and about 60 tons remained in one of the cyanide tanks, from which a portion of the values had been extracted, but which yet contained about 6 oz. of silver and 2.25 oz. of gold per ton. An analysis of these concentrates suggested the following approximate constitution: SiO_2, 17.64; CaO, 4.95; FeS_2, 52.73; Fe_2O_3, 25.31; Pb, not det.; Cu, none; As, none; total, 100.63 per cent.

Continued leaching of these concentrates, for about 21 days, with solutions of various strengths, removed about 75 per cent. of the remaining values, but with a large consumption of cyanide, when the extraction practically ceased. The material was thoroughly aërated by turning over with shovels, and one-half placed in an adjoining tank; both portions were again subjected to leaching with cyanide, but with no compensatory extraction.

In the above treatment of the concentrates, the first solutions drawn off were of a brilliant claret color; and, as no copper had been found in the analysis, the reason for the phenomenon was somewhat obscure. A repeated test of the concentrates again showed no copper, but an analysis of the material, precipitated from the colored solution by addition of acid, proved it to consist of cupric ferro-cyanide. Further consideration discovered the source of the copper in the residue left by evaporation in one of the solution-tanks; this had been taken up by the new solution and produced the color mentioned. Analysis of samples of freshly-made concentrates showed them to contain about 0.2 per cent. of copper; enough to assist nicely in the zinc-precipitation.

After various experiments upon the raw concentrates, it was found that if the material was ground to 100-mesh and agitated

for 32 hr., about 85 per cent. of the gold and 70 per cent. of the silver could be extracted, at an expense of about 6.5 lb. of cyanide per ton. This result promised a very fair return (especially in a case where no other procedure was available); and the system was put in effect in the mill, with the happy, and somewhat unusual, result that the mill-practice has yielded very much better returns than those obtained in the laboratory-experiments; a fact which is probably due to the rise in temperature produced during the grinding of the pulp.

The yield of concentrates from the mill was about 2 tons per day. For grinding so small a product, the purchase of a ball- or tube-mill was out of the question; so also was the use of one of the stamp-batteries for the purpose; and about the least expensive machine which could be thought of, which would have a sufficient capacity and which would probably fulfill the other requirements, was one of the old-style amalgamating pans, such as are used in silver-mills. A second-hand, 5-ft. pan, with wooden sides, was bought in Denver, installed in the mill, and arranged to discharge into each of two leaching-tanks belonging to the small cyanide-plant formerly mentioned. The pan was charged with about one ton of solution carrying 6 lb. of cyanide, 6 lb. of lime and 1.5 tons of raw concentrates; it was set in motion at the rate of about 75 rev. per min., and continued to grind for 8.5 hr.; about 2 lb. more of cyanide being added during the day, as the strength failed. At the end of the period, the material was found to be finely ground, and was discharged into the leaching-tank; it was also found that, at the close of the grinding, the temperature of the mass had risen about 40° F. above the outside temperature. A sample of the ground pulp was taken, filtered, washed and assayed, with the somewhat surprising result that an extraction was shown of 90.7 per cent. of the values.

Since the initial test, the operation has been carried on continuously with similar results; the only variation occurring when the grinding was affected by the clogging of the mullers by foreign matter. Since this occurrence, the concentrates have been passed through an 8-mesh screen, and no further difficulty has been noticed. After the grinder is discharged into the leaching-tank, and after the solution has settled to some extent, it is customary to cover the material in the tank

with dry middlings from the table, for the purpose of facilitating percolation; in this manner, filtration goes on with sufficient rapidity. When one leaching-tank is filled, it is continuously leached with cyanide solution, while the other tank is being filled; it is then washed and discharged through a bottom-discharge door. The capacity of each tank is about 15 tons; so that a tank has about 4 days of extra leaching after it is filled; it then has 3 days' washing before discharge. During a campaign of 6 months, in which time various grades of concentrates have been treated, the average extraction has been about 94 per cent. of the total values. The grinding and cyanide-plant require little attention, except in the matter of charging the grinder and discharging it, and the whole plant is easily operated, without additional cost, by the man who has charge of the tables and vanners. The pan consumes about 8 h.p., and, as stated, the total cyanide consumption is about 8 lb. per ton; adding the cost of 6 lb. of lime, the total cost of the treatment of the concentrates, including the values left in the tails, does not materially exceed $5 per ton.

The concentrates, as they are taken from the tables and vanners, are given as much chance as possible to aërate and oxidize, since it is found to be the case that partly oxidized material grinds more quickly and gives up its values with a notably less amount of cyanide. In the extraction of the final values from the old concentrates upon which tests were first made, it is found necessary to grind them for 2.5 hr. only. Muller-shoes last about 3 months; while the dies last much longer. The zinc-precipitation is good, and the solutions have not yet become too foul for use; most of the copper is slagged off in remelting the bullion.

In the first attempts at the use of the grinding-pan, it was the intention to make the pan work continuously, by overflowing and allowing the continual discharge of finely-ground ore-particles into the leaching-tank. It was found, however, that the rapid motion of the mullers carried coarse as well as fine to the surface, and the scheme was abandoned. It was next attempted to allow a continuous discharge through pipes set midway up the sides of the pan, sizing with cyanide solution by *spitzlutten*, and return of the coarse material to the grinder. This also was abandoned, and the discharge-opening at the

bottom of the pan was threaded and a valve put in. The charge in the pan is now ground, the valve connected with a discharge-pipe, the valve then opened and the contents of the pan discharged.

To show the grinding-efficiency of the pan, a series of sizing-tests were made upon concentrates as they came from the tables, and upon similar material after having been ground for 8 hr., as follows:

	Passed Through 100-mesh. Per Cent.	Passed Through 80-mesh. Per Cent.	Passed Through 60-mesh. Per Cent.	Coarser than 60-mesh. Per Cent.
Unground concentrates, . . .	51.08	14.7	6.20	28.02
Ground concentrates, ...	82.73	9.20	4.47	3.60

In crushing to the above degree of fineness, the material seemed to be fairly amenable to cyanidation, while not too fine for good filtration; so that 8-hr. grinding is continued.

Comparison of American and Foreign Rail-Specifications, With a Proposed Standard Specification to Cover American Rails Rolled for Export.

BY ALBERT LADD COLBY, NEW YORK, N. Y.

(London Meeting, July, 1906)

TABLE OF CONTENTS.

I. INTRODUCTION.

A GLANCE through the Bibliography appended to this paper will show that the *Transactions* of this Institute contain what virtually constitutes a history of the development of the manufacture of steel rails in this country and abroad. It is, therefore, fitting that this paper, bringing one branch of this important subject down to date, should be here presented, and it is also appropriate to select this joint meeting of our Society with the Iron and Steel Institute to compare the specifications covering the rails produced in these two countries, and to propose a standard specification to govern the manufacture of American rails for export.

During the last ten years the production of Bessemer steel rails in the United States and Great Britain has amounted to 32,128,720 gross tons. Nearly 19,000,000 tons have been rolled in the past five years. The next five years will see an even larger output, owing partly to the extension of railway projects in the Orient, in South and Central America, in Mexico, and in Australia and other British colonies.

Of late years, the American rail-mills have so largely increased their daily output, that trade-conditions have often made possible the delivery of American rails to foreign ports.

The tenders submitted by American mills, however, have always been accompanied by requests for changes in the foreign specifications, chiefly in the reduction of the amount of testing required; and it has been only natural, especially at first, that the engineer, whose specification had always been accepted without question by British mills, should interpret American tenders as an evidence that an inferior rail would be furnished

his client if the concessions were granted. Often after a considerable discussion, the specifications have been modified, in some instances on the representation of independent American engineers, who make rail-inspection a specialty, that the requested modifications or omissions in the requirements would not reduce the quality of the rails furnished.

Although the delivery of many tons of American rails to foreign ports has proved such to be the case, there has been no general acknowledgment, on the part of foreign engineers, that their rail-specifications must be modified if American tenders are particularly desired; instead, the necessary concessions are still a matter for individual adjustment, and, at best, the modified specification still leaves the interpretation of certain requirements in such doubt that the American mills accept orders for rails for export, with the cost due to inspection, and the possibility of a reduced output, as very uncertain factors.

The object of this paper is to clear up this matter for the best interests of all concerned. A sufficient tonnage of American rails is now in service abroad, especially in the British colonies, to establish the fact that in no respect do they give a less satisfactory service than the product of British mills.

In the past five years much has been accomplished in both the United States and Great Britain toward the standardization of the requirements governing this large item in the tonnage of steel produced in both countries. In this paper the importance of what has been thus accomplished is duly recognized, and the requirements of the current standard and individual rail-specifications now governing the rolling of rails in both countries are duly compared, so as to show that the fewer and more definite requirements which now govern the large tonnage of rails rolled for American railroads make it unnecessary, in order to obtain the same quality of product, to hamper the American methods of manufacture by the more elaborate requirements of foreign specifications.

The reasons why foreign rail-specifications cannot be applied to American mills are stated plainly, and a standard rail-specification is proposed which is particularly applicable to the manufacture of rails rolled in America for export: its requirements can be met by all the American mills, so that the possibility of competition is assured; the specification includes all the checks

on the manufacturer of rails essential in the present state of the art in American mills; the requirements are definite; all ambiguous and unnecessary clauses are omitted, the idea being that a specification should be a contract strictly lived up to by both parties concerned, and having for its object the delivery, without unnecessary expense to either party, of the best possible material which can be furnished commercially for the purpose intended.

An explanation should be given as to just why the output of American rail-mills has of late years so materially increased, so as to show that this change has not reduced the quality of the product.

The gradual increase in the daily output of American rail-mills during the last decade is not due to an increase in the speed of the rail-train, prompted by a reckless desire for increased tonnage, independent of the quality, strength and finish of the product, but rather to radical improvements resulting in a much better balance between the producing- and finishing-ends of the mill. In the first place, the finishing-train is used only for rolling rails, hence there are few delays due to roll-changes; furthermore, the rail-train is now kept much busier; the bars are brought more quickly and continuously to the rail-train by the use of tables with rapidly revolving rollers; the blooming-train is seldom idle, owing to the ample heating-capacity of the modern soaking-pits and the quick stripping and handling of ingots; sufficient air-pressure is available to blow all the converters at once, if necessary; and finally, there is always an excess supply of melted pig-iron for the converters, drawn either direct from adjacent blast-furnaces, or from storage-receivers known as "mixers," which are sometimes supplemented with iron remelted in cupolas.

In times past the converting- and blooming-departments have not equaled the capacity of the rail-train, but by the improvements briefly outlined above, the lost-time intervals have been materially cut down. One American rail-mill, without change in the speed of its rail-train, has thus increased its output of rails from 1,000 to 1,800 tons per day of 24 hours.

That the maximum capacity of its finishing-train has not yet been reached, without increasing its speed, may be seen by the following calculation. With the finishing-roll 26 in. in di-

ameter, running at a speed of 95 rev. per min., its product, if a continuous bar were rolled, would, if cut up into 33-ft. lengths, make no less than 28,221 rails per 24. hr., or 11,087 gross tons of 80-lb. 33-ft. rails on two shifts; of course this is a theoretical output, but it proves the error of the usual criticism that the present output of American rail-mills is due to very rapid rolling.

In the following comparative discussion of the American and British specifications which govern the rails produced to-day, the requirements have been classified under the headings given in the table of contents, and this same classification has also been used in the proposed standard specification found at the end of this paper.

II. Process of Manufacture.

1. *American Specifications.*

The specifications of American railroads, as well as those known as "the Manufacturers' Specifications," which latter govern, when no specification is submitted by the railroad, both contain a general provision, that the entire process of manufacture and testing shall be in accordance with the best current practice, and both mention certain precautions in the process with particular reference to obtaining sound rails.

2. *Foreign Specifications.*

In some foreign rail-specifications there are clauses governing the process of manufacture which should be omitted when applying the specifications to American practice. While all American makers should readily comply with a requirement that, in their tender, they must mention the character of the materials and the process they propose to use, their sources of supply are such that clauses worded as follows should be waived: "Best hematite pig-iron," or "No. 1 Cumberland hematite pig-iron must be used." "The iron used to be made from the best English or Spanish hematite ore." "Charcoal spiegeleisen shall be used." As "direct metal" is almost entirely used in American mills, the requirement that "the iron be melted in air furnaces, or cupolas, before being run into the converters" should be omitted.

In reference to ingots, some specifications require that "each ingot shall be of ample length and weight to allow for cutting off unsound parts from each end of the rail to reduce it to the finished length;" others require that the ingot shall measure at least 10 in. at the top and 12 in. at the bottom, or at least 12 in. and 14 in., respectively. In American practice all rail-ingots are of ample size for discards; in fact, from 4 to 6 rails of the heaviest sections are rolled from each ingot. A clause such as the following is unnecessary and impracticable: "The surfaces of the ingots are to be smooth and clean; any inequalities produced by the ingot-molds must be removed by chisel. No ingots containing any defect will be accepted, and no repairs to defective ingots will be permitted." The judgment of the manufacturer as to the disposition of the ingots should govern, limited, however, by the requirement that he must be in accord with the best current practice. He would gain nothing by attempting to repair defective ingots, as to make such repairs he would first have to allow the ingot to become cold.

As no American rail-mill allows ingots to become cold, all rails, containing surface-defects due to superficial defects in ingots, are rejected before being submitted to the inspector.

III. Chemical Properties.

1. *Chemical Composition.*

(*a*) *American Specifications.*—In American rail-specifications the carbon and manganese requirements increase with the weight of the section. Silicon is usually specified "not over 0.20 per cent.," which limit, with American acid-Bessemer blowing, might as well be omitted, unless silicon is purposely added, to meet special requirements. Sulphur is only occasionally specified, the upper limit being 0.075 per cent.; this is essential, as a higher sulphur-content causes red-shortness, with its attendant evils in the finish of the rail.

The limits in Table I., quoted from the manufacturers' standard specifications, govern, with but slight modifications in a few cases, by far the largest tonnage of rails delivered to-day to American railroads.

TABLE I.—*Chemistry of American Rail-Specifications.*

Weight of Rail Per Yard.	Carbon.	Manganese.	Silicon. Not Over.	Phosphorus. Not Over.
Pounds.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
50–59	0.35–0.45	0.70–1.00	0.20	0.10
60–69	0.38–0.48	0.70–1.00	0.20	0.10
70–79	0.45–0.55	0.75–1.05	0.20	0.10
80–89	0.48–0.58	0.80–1.10	0.20	0.10
90–100	0.50–0.60	0.80–1.10	0.20	0.10

In rails of 70 lb. and over, the range of carbon above specified is 5 points (0.05 per cent.) higher than standard practice in 1900.

The recent specifications of two committees of engineers recommend still higher carbons than given in Table I., lower the upper limit of manganese 0.05 per cent., and suggest a reduction of the phosphorus to 0.085 per cent.

No data have been presented proving that rails of the carbons specified in Table I. have, because of a content of 0.10 per cent. of phosphorus, failed or proved brittle after short service. Considerable data have been published showing that rails made under specifications allowing 0.10 per cent. of phosphorus have given excellent service. The vast tonnage of rails now in track and showing an excellent record, which, if analyzed, would show from 0.090 to 0.110 per cent. of phosphorus, weakens the plea for a reduction in the present guaranteed phosphorus of steel rails.

To obtain a harder steel, or a better rail, by an increase in carbon and a reduction in phosphorus is theoretically ideal. To obtain steel rails with a guaranteed phosphorus-content lower than 0.10 per cent. from the majority of American rail-makers is now impracticable. The one Eastern rail-mill using pig-iron made entirely from Cuban ores can, without extra cost, make steel of a somewhat lower guaranteed phosphorus than the seven Western rail-mills using iron made from native ores. By a careful selection of ores, some of these latter mills could make a small tonnage of steel rails of a guaranteed phosphorus-content of 0.085 per cent.; but the extra cost and other trade-conditions make this suggestion impracticable, and existing data as to wear do not warrant establishing a phosphorus-guarantee below 0.10 per cent.

(*b*) *Foreign Specifications.*—Contrary to American specifications, there is very seldom in foreign rail-specifications any change in the carbon and manganese with an increase in the weight of section. The maximum carbon is 0.50 per cent., usually with an allowable range of 15 points, except in the British standard specification for tramway-rails of from 90 to 116 lb. per yd., where the carbon-range is from 0.40 to 0.55 per cent.

Table II. also shows that the maximum manganese specified is 1.00 per cent., with allowable range of 30, but sometimes 20 points.

There is considerable variation in the maximum silicon specified, the figures being 0.06, 0.07, 0.08, 0.10 and occasionally 0.15 and 0.20 per cent.

With sulphur, the maximum is either 0.080, 0.070 or 0.060 per cent.

The American mills have no objection to making the softer rails preferred by foreign engineers, and consequent upon maximum limits of 0.50 per cent. of carbon, 1.00 per cent. of manganese and 0.080 per cent. of silicon. They need a range of only 10 points in carbon, although often given, as above stated, a limit of 0.35 to 0.50 per cent. They want a range of 20 points in manganese, either from 0.80 to 1.00 per cent., or from 0.70 to 0.90 per cent., but they object to such low manganese as 0.40 per cent., which is occasionally specified. In reference to sulphur, a fair maximum would be 0.070 per cent.

As to phosphorus, the requirement in foreign specifications of older date, that every heat of rail-steel shall be analyzed for phosphorus and arsenic, and stipulating that together they must not exceed 0.60 per cent., has become obsolete. Arsenic should not be classed with phosphorus in its effect upon the physical qualities of steel, and furthermore, there is no rapid method by which phosphorus and all the arsenic can be determined in steel. Most of the current foreign rail-specifications limit the phosphorus to 0.080 or 0.075 per cent., although some published analyses of British rails show a higher phosphorus-content than 0.080 per cent.

In discussing the American specifications, it was stated that none of the American mills, except the one using foreign ores entirely, would guarantee their output of steel rails under

TABLE II.—*Chemistry of Foreign Rail-Specifications, Arranged According to Weight of Section.*

50-59 Lb.			60-69 Lb.			70-79 Lb.			80-89 Lb.			90-100 Lb.			Silicon. Not Over.	Phos-phorus. Not Over.	Sulphur Not Over.
Weight.	Carbon.	Man-ganese.	Weight.	Carbon.	Man-ganese.	Weight.	Carbon.	Man-ganese.	Weight.	Carbon.	Man-ganese.	Weight.	Carbon.	Man-ganese.			
	Per Cent.	Per Cent.		Per Cent.	Per Cent.		Per Cent.	Per Cent.		Per Cent.	Per Cent.		Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.
...	...	...	60	0.35–0.42	0.75–1 00	75	0.35–0.42	0.75–1.00	80	0.35–0.42	0.75–1.00	100	0.35–0.42	0.75–1.00	0.10	0.080	0.060
...	...	...	65	0.35–0.42	0.75–1.00	...	...	...	85	0 35–0.42	0.75–1.00	...	...	...	0.10	0.080	0.060
50	0.30–0.45	...	...	...	...	75	0.30–0.45	...	85	0.35–0.50	Max. 0.80	...	...	...	0 06	0.080	0.060
...	...	...	...	...	...	75	0.30–0.50	Max. 0.80	82	0 35–0.50	Max. 0.80	...	...	...	0.06	0.080	0.060
30	0.30–0.45	0.80	...	...	...	...	...	...	...	...	...	...	...	...	0.06	0.080	0.060
50	0.30–0.45	0.80	...	...	...	...	...	...	...	...	...	...	...	...	0.06	0.080	0.060
...	...	...	...	...	...	...	...	...	...	...	...	95	0.40–0.50	0.85–0.95	0.10	0.080	0.080
...	...	...	...	...	...	...	...	...	85	0.35–0.45	0.70–0.95	...	...	...	0 10	0.080	0.070
...	...	...	...	...	...	72.8	0.40–0.50	Max. 1.00	85	0.40–0.50	Max. 1.00	...	...	...	0 08	0.075	0 070
...	...	...	60	0.30–0.45	Max. 1.00	...	...	...	...	...	...	...	...	...	0.10	0.075	0.070
...	...	...	...	...	...	...	...	...	...	...	...	100	0.40–0.50	0.80–1.00	0.08	0.075	0.070
...	...	...	...	...	...	...	...	...	83	0.40–0.50	0.70–1.10	...	...	...	0 15	0.080	0.080
...	...	...	60	0.38–0.48	0.70–1.00	...	...	...	...	...	...	...	...	...	0.20	0.100	...
...	...	...	...	...	...	...	...	...	80	0.42 Min	Max. 1.10	100	0.45 Min.	Max. 1.10	0.20	0.100	0.080
...	...	...	...	...	...	...	...	...	80	0.40–0.50	0.80–1.00	...	...	...	0 15	0.080	0.080
...	...	...	...	...	...	...	...	...	...	...	...	90	0.40–0.55	0.70–1.00	0.10	0 080	0.080
...	...	...	...	...	...	...	...	...	...	...	...	116	0.40–0.55	0 70–1.00	0.10	0 080	0.080
...	...	...	60	0.35–0.50	0.70–1.00	70	0.35–0.50	0.70–1.00	80	0.35–0.50	0 70–1.00	90	0.35–0.50	0.70–1.00	0.10	0.075	0.080
...	...	...	65	0.35–0.50	0.70–1.00	75	0.35–0.50	0.70–1.00	85	0.35–0.50	0.70–1.00	95	0.35–0.50	0.70–1.00	0 10	0.075	0.080
...	...	...	...	...	...	...	...	...	...	...	...	100	0.35–0.50	0.70–1.00	0.10	0.075	0.080
20	0.35–0.50	0.70–1.00	...	...	...	...	...	...	...	...	...	...	...	...	0.10	0.070	0.070
40	0.35–0.50	0.70–1.00	60	0.35–0.50	0.70–1.00	70	0.35–0.50	0.70–1.00	80	0.35–0.50	0.70–1.10	90	0.35–0.50	0.70–1.00	0 10	0 070	0.070
50	0.35–0.50	0.70–1.00	65	0.35–0.50	0.70–1.00	75	0.35–0.50	0.70–1.00	85	0.35–0.50	0.70–1.10	95	0.35–0.50	0.70–1 00	0.10	0.070	0.070
55	0.35–0.50	0.70–1.00	...	...	...	...	...	...	...	...	...	100	0.35–0 50	0.70–1.00	0.10	0.070	0.070
...	...	...	60	0.30–0.45	Max 1 00	...	...	...	...	...	...	...	...	...	0 07	0 075	0 070
...	...	...	...	...	...	...	...	...	80	0.30–0.45	Max. 1.00	...	...	...	0.07	0 075	0.070
...	...	...	...	...	...	...	...	...	...	...	...	93	0.40–0.50	0.50–0.80	0.12	0.060	0.060
...	...	...	60	0.30–0.45	Max. 1.00	...	...	...	...	...	...	...	...	...	0 07	0 075	0.070
...	...	...	...	...	...	...	...	...	...	...	...	92	0.35–0.50	0.75–1.00	0.10	0.080	0.080
...	...	...	...	...	...	...	...	...	87	0.40–0.50	0.70–1.10	...	...	...	0.15	0.080	0.080
...	...	...	60	0.38–0.48	0.70–1.00	...	...	...	...	...	...	...	...	...	0.20	0.100	...
...	...	...	...	...	...	70	0.40–0.50	0.75–1.05	...	...	...	...	...	...	0.20	0.100	0.070
...	...	...	...	...	...	75	0.40–0.50	0.75–1 05	...	...	...	...	...	...	0.20	0.100	0.070
...	...	...	60	0.35–0.50	0.75–1.00	...	...	...	...	...	...	...	...	...	0.10	0.080	0 100
...	...	...	60	0.38–0.48	0.70–1.00	...	...	...	...	...	...	...	...	...	0.20	0.100	...
...	...	...	60	0.35–0.45	Max. 0.90	...	...	...	...	...	...	...	...	...	...	0.100	...
...	...	...	...	...	...	...	...	...	80	0.45–0.50	0.85–0.95	90	0.45–0.50	0 85–0 95	0.06	0.080	0.080
50	0.30–0.45	Max. 0.80	60	0.35–0.50	Max. 0.80	...	...	...	...	...	...	...	...	...	0.06	0 080	0.060
...	...	...	...	...	...	...	...	...	86	0.35–0.50	0.75–1.00	...	...	...	0.10	0.075	0 080
...	...	...	...	...	...	...	...	...	85	0.35–0.45	0.85–0.95	...	...	...	0.10	0.080	0.080
...	...	...	...	...	...	...	...	...	80	0.42 Min	Max. 1.10	100	0.45 Min.	Max. 1.10	0.20	0.100	0.080

0.10 per cent. of phosphorus. It is, therefore, evident that, to obtain tenders from several American mills, the foreign engineer should modify his specified phosphorus to 0.10 per cent. from the present limits of 0.075 or 0.080 per cent., which English makers of acid-Bessemer steel rails can meet without extra cost, having the Cumberland and Spanish hematites, of lower phosphorus than the American ores, as a basis for their ore-mixtures. With the low carbon specified, this change, resulting in free competition, is not as serious as might at first appear.

The engineer knows that even the steel-maker admits that phosphorus is the most undesirable constituent of steel. He knows that, in general, when a steel is liable to sudden and frequent shock, it should be "low in phosphorus." He must bear in mind, however, that careful heating and lower finishing-temperatures easily counteract a difference of 0.020 per cent. in phosphorus; moreover, it is universally conceded that the evil effects of phosphorus increase with the carbon; that the foreign engineer's rail-specifications call for a lower range of carbon than is usually specified in America; that the higher-carbon American rails with their 0.10 per cent. of phosphorus withstand the shock of the drop-test, and that it has not been proved that any difficulties with breakage on American roads or on foreign roads laid with American rails have been attributed to the presence of 0.10 per cent. instead 0.08 per cent. of phosphorus in the rails. Nor in foreign service have American rails been proved more brittle than those of British make.

Finally, he should bear in mind that this modification in phosphorus is by no means a new suggestion, as it has often been allowed; a fact emphasized by the official statistics, which show that in the last ten years 2,262,451 gross tons of rails have been exported from the United States, and that therefore, abroad as well as in America, there is a considerable tonnage of rails containing a little less than 0.10 per cent. of phosphorus, now in track, and meeting the service demanded.

Basic open-hearth, or even basic Bessemer rails with their very much lower phosphorus-content—if other hardening-constituents were present in the right proportion and other even more important details of manufacture were properly carried out—would perhaps give better and longer service; but under

present conditions, the vast majority of rails rolled in England and America for some time to come will be made of acid-Bessemer steel.

Under these commercial conditions, and in view of the facts presented, and the other requirements of his specification, which in themselves amply protect his client, is the engineer warranted in restricting competition by refusing to change his phosphorus limit to 0.10 per cent. so as to include current American practice?

2. *Chemical Analysis.*

(*a*) *American Specifications.*—In American specifications, the manufacturer is required to furnish the inspector daily with carbon-determinations from each blow, and a complete chemical analysis every 12 or 24 hr., representing the average percentage of manganese, phosphorus, sulphur and silicon present.

All these analyses are made on drillings taken from the test-ingot, cast when pouring the heat, and check-analyses on rails are not required.

(*b*) *Foreign Specifications.*—The demands on the steel-works laboratory and the possible delays resulting from independent check-analyses are of such importance to the American rail-mill, when the foreign requirements are applied to American output, that the necessity for such frequent analyses and the emphasis given to checking results will be discussed somewhat in detail.

(*c*) *Number of Analyses Required.*—In some foreign rail-specifications, the maker is asked to furnish complete chemical analyses more frequently that can possibly be complied with, when the requirements are applied to American practice; even with the facilities of the modern rapid analytical methods, the conscientious efforts of steel-works chemists fail to meet the demands made on them.

It is unnecessary in American practice to obtain such frequent checks, for in the present state of the art, the "direct metal," as delivered to the Bessemer vessel, is smelted from a uniform mixture of ores, so that the variation in phosphorus is small, and the "blowing" is so constant and regular that temperatures vary within narrow limits, and hence the silicon in the steel is practically constant. As to reports on carbon and manganese, the maker is willing to furnish them with every

blow of steel; also phosphorus-determinations on six blows of steel every 12 hr., and sulphur and silicon on an average sample of each 12 hours' rolling.

(*d*) *Sample for Analysis.*—Drilling from a test-ingot, cast during pouring, is the only practicable method by which an identified average sample of a heat of steel rolled into rails can be obtained, and American mills refuse to allow analyses of drillings from rails or tensile tests to govern the decision of the inspector as to the chemical composition of a heat of rails, because such samples are not representative.

It is well known to the consulting engineer and his representatives at the mill, as well as to the manufacturer, that there is an unavoidable variation in the composition of the different rails from each blow, as well as those from each ingot, a difference between the two ends of a rail, and, in point of fact, unavoidable variations even in the different portions of the cross-section. It is, therefore, unreasonable to specify that check-analyses shall be made on rails, and that the finished rails shall be piled into 500-ton lots and not shipped until an analysis on one piece of rail is reported by some independent and probably far-distant chemist, who has been selected by the engineer.

If the average chemical composition of a piece of rail is to be determined, there is only one correct way of obtaining an average sample—namely, to machine dry clean chips from the entire cross-section of the rail for a uniform depth, weigh the entire sample, dissolve it all in acid, dilute the solution to a known volume and measure off aliquot portions for the determination of manganese, silicon and phosphorus; again machine the entire cross-section for a smaller depth, and use all the chips for a gravimetric determination of carbon, preferably by combustion in oxygen; again machine the cross-section and use all of the sample for the determination of sulphur.

No chemist can dispute the above statement, and yet many steel chemists will admit that they have been requested to analyze drillings from rails which they knew did not truly represent the piece of rail selected as a sample, much less the 500 tons which the inspector was privileged to accept or reject on the basis of the reported analysis.

(*e*) *Check-Analyses Impracticable.*—The laboratories of the large American steel-works are in charge of competent chem-

ists using modern methods. Their work is regularly checked by American customers buying merchant steels on guaranteed analyses. Standard samples of pig-iron and steel, which have been carefully analyzed, are now furnished at low cost by the Bureau of Standards at Washington, for the purpose of checking chemists' results. A steel-works chemist, by virtue of the many similar determinations he is daily required to make, becomes a specialist in his line, and it is no injustice to claim that his results are at least as accurate as those of independent chemists, whose analytical work usually covers a much wider field.

For these reasons, the American rail-maker, who willingly furnishes daily a sufficient number of analyses to form a correct opinion of the chemical composition of his product, is justified in declining to comply with the many forms of check-analyses found in foreign rail-specifications, if his product may be thus rejected without appeal, and when it is specified that these independent analyses may be made "as often as desired" and "at the expense of the contractor."

It is impracticable in American practice to re-handle the finished rails, sorting them out in piles of 200 or 500 tons, and it is entirely unjust, as will be shown in the discussion immediately following, for such lots to be rejected solely on the analysis of a single rail.

(*f*) *Rejection Solely on Variation in Composition.*—The provisions of foreign rail-specifications for the rejection of rails solely on the basis of a variation from the prescribed chemistry are too strictly drawn, and the suggestion as to check-analyses is impracticable.

In studying the prompt and frequent analyses of the rail-steel made on each turn, furnished by the manufacturer, it is but reasonable that the inspector should be given the authority to exercise some judgment, when the analyses show only a slight and immaterial variation from the specified limits. By so doing, he is not jeopardizing the interests of the purchaser, but simply acknowledging that accuracy within 0.005 per cent. in phosphorus, sulphur and silicon, and 0.01 per cent. in carbon and manganese, is not uniformly attainable with the rapid methods of analysis necessarily employed; and he is further supported in a sensible tolerance in this regard by the knowl-

edge that no such small variations from the prescribed limits in chemistry can possibly affect the life, the safety or the wear of the rails when put in service.

Furthermore, no prescribed methods for chemical analyses should form a part of a specification. Entire uniformity in the details of the various analytical methods used in steel-works laboratories will probably never be realized, nor is it a necessity in obtaining accurate chemical analyses.

IV. Physical Properties.

1. *Drop-Test and Drop-Testing Machine.*

(*a*) *American Specifications.*—As one of the results of the standardization of rail-specifications in America, the American rail-mills are equipped with a drop-testing machine, uniform in its essential features.

The tup weighs 2,000 lb.; the radius of its striking-face is 5 in. The weight of the anvil-block is amply sufficient to secure rigidity; and furthermore it rests on a solid masonry foundation. The supports for the rails have a 5-in. radius, are set 3 ft. apart, and are firmly secured to the anvil-block. The height to which the tup is raised is measured in even feet, by a fixed vertical gauge.

The drop-test is made on a piece of rail, from 4 to 6 ft. long, cut at the hot saws, from the end of one rail from every fifth heat. The piece is stamped with the blow-number and placed on skids to cool. When cool and ready for test, it is placed head upwards on the supports, and the various sections must withstand, without fracture, one blow of the 2,000-lb. tup, from the heights specified. The report of this drop-test includes a record of the atmospheric temperature at the time the tests were made.

None of the American specifications require a second blow from the tup, and, with but one or two exceptions, no deflection-limits are specified.

In case the rail-butt, selected to represent the heat, as above described, should break when subjected to the drop-test, two additional drop-tests are made on pieces of rail or on rail-butts cut from two other rails from the same blow of steel; if either of these latter tests fail, all the rails of the blow which they

represent are rejected, but if both of these additional test-pieces meet the requirements, all the rails of the blow which they represent will be accepted. If a heat of rails is thus rejected, the blows next preceding and succeeding the rejected blow are similarly tested, and if the rails of either of these blows are rejected, provision is made for similarly testing adjacent blows until the entire group of five blows is tested, if necessary. Each blow is accepted or rejected, according to the result of the drop-test on the test-rails representing the blow.

Table III. gives the heights of drop required in the specifications governing the vast majority of the tonnage of domestic rails delivered to American railroads. The chemical requirements are included for comparison; the "foot-pounds" noted are simply the product of the weight into the height, without regarding the energy of the tup just before striking. The comparative resistance to impact of rails, of a certain weight per yard, is influenced by the section. As more than 80 per cent. of the rails now rolled for American railroads are the standard "T" sections recommended in 1893 by the American Society of Civil Engineers, the weight and the chief measurements of these standard sections are also included in Table III. so as to aid in comparing the American drop-test on "T" rails of certain weights and dimensions, with that required for the wide variety of sections included in foreign specifications.

(*b*) *Foreign Specifications.*—A tabular comparison of the drop-test, as it is found in foreign rail-specifications, would only emphasize the diversity of the requirements specified. Since the specifications are based on British practice, the number of tests usually required should be reduced when applied to American practice, so as not to retard the output; this concession is reasonable because the inspector is given the carbon- and manganese-content of each heat before the rails are rolled, and the continuous methods characteristic of American rail-mills make but slight differences in the finishing-operations. With the chemical composition already known, a drop-test on each heat is therefore not necessary, and with the production of more than 125 heats per 24 hr., this requirement operates as a hardship on the manufacturer as well as on the inspector.

A few specifications make no allowance for re-test, in case the piece cut from the end of one rail of the heat fails under the

TABLE III.—*Drop-Tests of American Rail-Specifications.*

Weights per Yard for Chemical Requirements	50-59 Lb. per Yd.				60-69 Lb. per Yd.		70-79 Lb. per Yd.		80-89 Lb. per Yd.		90-100 Lb. per Yd.	
Carbon	0.35-0.45 per cent.				0.38-0.48 per cent.		0.45-0.55 per cent.		0.48-0.58 per cent.		0.50-0.60 per cent.	
Manganese	0.70-1.00 per cent.				0.70-1.00 per cent.		0.75-1.05 per cent.		0.80-1.10 per cent.		0.80-1.10 per cent.	
Silicon	Not over 0.20 per cent.				Not over 0.20 per cent.		Not over 0.20 per cent.		Not over 0.20 per cent.		Not over 0.20 per cent.	
Phosphorus	Not over 0.10 per cent.				Not over 0.10 per cent.		Not over 0.10 per cent.		Not over 0.10 per cent.		Not over 0.10 per cent.	
Weights per Yard for Drop-Tests	45-55 Lb. per Yd.				55-65 Lb. per Yd.		65-75 Lb. per Yd.		75-85 Lb. per Yd.		85-100 Lb. per Yd.	
A. S. C. E. Standard "T" Sections.												
Weight per Yard	40	45	50	55	60	65	70	75	80	85	90	100
Height	$3\frac{1}{2}$ In.	$3\frac{11}{16}$ In.	$3\frac{7}{8}$ In.	$4\frac{11}{16}$ In.	$4\frac{1}{4}$ In.	$4\frac{7}{16}$ In.	$4\frac{5}{8}$ In.	$4\frac{13}{16}$ In.	5 In.	$5\frac{3}{16}$ In.	$5\frac{3}{8}$ In.	$5\frac{3}{4}$ In.
Width of Base	$3\frac{1}{2}$ In.	$3\frac{11}{16}$ in.	$3\frac{7}{8}$ In.	$4\frac{1}{16}$ In.	$4\frac{1}{4}$ In.	$4\frac{7}{16}$ In.	$4\frac{5}{8}$ In.	$4\frac{13}{16}$ In.	5 In.	$5\frac{3}{16}$ In.	$5\frac{3}{8}$ In.	$5\frac{3}{4}$ In.
Width of Head	$1\frac{7}{8}$ In.	$2\frac{1}{8}$ In.	$2\frac{1}{8}$ In.	$2\frac{1}{4}$ In.	$2\frac{3}{8}$ In.	$2\frac{13}{32}$ In.	$2\frac{7}{16}$ In.	$2\frac{15}{16}$ In.	$2\frac{1}{2}$ In.	$2\frac{9}{16}$ In.	$2\frac{5}{8}$ In.	$2\frac{3}{4}$ In.
Thickness of Web	$\frac{25}{64}$ In.	$\frac{27}{64}$ In.	$\frac{7}{16}$ In.	$\frac{15}{32}$ In.	$\frac{31}{64}$ In.	$\frac{1}{2}$ In.	$\frac{17}{32}$ In.	$\frac{17}{32}$ In.	$\frac{9}{16}$ In.	$\frac{9}{16}$ In.	$\frac{9}{16}$ In.	$\frac{9}{16}$ In.
Height of Drop	14 Ft.				15 Ft.		16 Ft.		17 Ft.		18 Ft.	
Equivalent to Foot-Pounds	28,000				30,000		32,000		34,000		36,000	

impact-test. Usually, however, privilege is given to submit tests cut from two other rails from the heat, and the heat is rejected if two out of three tests fail.

As the specified distance between supports is never over 4 ft., and usually not over 3.5 ft., it simply makes the manufacturer unnecessarily consign good metal to scrap, for the engineer to specify the test of pieces over 5 ft. long or of a full-length rail, as is sometimes done.

When the drop-test accepts or rejects the product by heats, the further check of one test cut from every 100 or 200 rails, as is sometimes specified, seems unnecessary.

There is one form of foreign drop-test which acts as a hardship on the American mills if made as specified on "one rail-butt from each heat." The test requires that a piece of rail, with 4 ft. between supports, shall be given two blows with a 1-ton tup from sufficient heights to bend the rail to an angle of 110° (that is, through an angle of 70°); the specified carbon is from 0.35 to 0.42 per cent. It seems illogical to specify this fixed angular deflection for all sections and weights of rails, and it is questionable whether this requirement can be obtained from the heavier sections under the conditions specified.

Specified drop-tests, such as the following, are entirely impracticable when applied to American output. After the regular drop-test, it is specified that "the rails are also to be tested until broken, the heights of fall and number of blows being recorded;" also, "butts of rails will be subjected to drop-tests until broken, and if the metal fails to show 4 per cent. of elongation before breaking, the whole lot of rails made from the heat will be rejected."

(c) *Limits in Deflection.*—As a general rule, an American rail will not show quite as much deflection under impact as a similar section of British make. One reason is that the carbon specified by foreign engineers is so much lower than American practice that the natural tendency of the manufacturers is to approach the upper limit of the permissible range, as they are not accustomed to making rail-steel below 0.40 per cent. of carbon; another, that the finishing-temperature also differs somewhat from that of British mills.

American mills do not hesitate to guarantee that their rails, rolled to the required foreign section and to the chemistry speci-

fied, will show entire freedom from brittleness under an impact-test made by their drop-test machine, if measured by the ability of the rails of various weights to withstand one blow, equivalent to the foot-pounds contained in Table III. Since a rail, if brittle, will fail on the first blow, a continuance of the testing, as well as a specified range of deflection between narrow limits, seems unnecessary.

The drop-test is a check on brittleness; the fact that a rail of a certain section as delivered from an American mill will not show as much deflection as a similar rail of British make, is not proof that the American rail will prove less safe, and as far as resistance to wear is concerned, there is no evidence that, under like conditions, the American rail is giving an inferior service; in fact, there are indications that the stiffer rail is not unsafe and gives better wear than the softer rail which showed, when inspected at the mill, a greater deflection under impact. Furthermore, the stiffer rail insures a longer life to the rolling-stock.

Deflections are therefore omitted in the proposed standard specifications appended to this paper.

2. *Tensile Test.*

This requirement is so frequently found in foreign rail-specifications, that its actual value, as well as the difficulties encountered in applying it to American practice, to the extent that it is usually specified, will be discussed somewhat in detail.

(*a*) *American Specifications.*—None of the American rail-specifications include a specified tensile strength, elongation or contraction of area, because the data thus accumulated are not considered of any practical value to the purchaser.

(*b*) *Foreign Specifications.*—As a result, the American rail-mills are not especially equipped rapidly to machine tensile specimens from the rails and determine promptly the tensile strength and elongation. They therefore object, in general, to any tensile requirements, and especially to specifications in which a tensile test is required so frequently as to render it impossible to keep the testing up to their tonnage output, no matter how well they might equip themselves to meet this requirement, found only in foreign rail-specifications.

It is not just to the manufacturer to require him to furnish

rails within certain limits in tensile strength, elongation and, sometimes, reduction of area, when the specification also requires the steel to be within a certain range in carbon and manganese, and of a certain limit in silicon, and when the maker willingly furnishes prompt determinations of carbon and manganese in every heat of steel rolled into rails, and also meets a specified drop-test.

(c) *Test-Pieces are not Representative.*—The actual value of this requirement is doubtful, because the tensile specimen, at the best, but imperfectly represents the strength of the steel in the rail selected for test; furthermore, no data have been presented proving that the results obtained on tensile specimens, as cut from rails, bear any relation to the wear of the rails in service.

(d) *Specified Tensile Strength is Affected by the Section.*—With the same specified range in chemical composition for rails weighing 20 lb. to the yd. and those of 100 lb. to the yd., it is manifestly inconsistent to require the steel to come within the same limits in tensile strength and meet the same minimum elongation, and thus take no cognizance of the more rapid cooling of the lighter sections.

Table IV. gives examples of foreign rail-specifications where the same tensile strength and elongation are required for a wide range of sections. The injustice of this disregard of the effect of the section is emphasized when, as noted, these same specifications permit the rejection of all the rails of a blow, if the tensile tests from one to three rails fail to meet the requirements.

(e) *Specified Tensile Strength is Often Inconsistent with Specified Chemistry.*—It has been stated above, that it is not just to hold the manufacturer to a specified chemistry, and in addition require him to meet definite limits in tensile strength, elongation and contraction of area. In some foreign rail-specifications this injustice is, if possible, emphasized by the engineer requiring physical properties which cannot be met at all, if the manufacturer adheres to the specified chemistry. The required elongation is often too high for even the minimum tensile strength specified, to say nothing of the maximum tensile strength, which should, in no event, be included in rail-specifications.

TABLE IV.—*Quotations from Foreign Rail-Specifications in which the Same Tensile Strength and Elongation are Specified Independent of the Weight of the Rail Section.*

Weight of Rail.	Tensile Strength, Per Sq. In.		Elongation.	Failure of these Tensile Requirements Causes Rejection.
Pounds Per Yard.	Tons.	Pounds.	Per Cent.	
90 to 116	Not less than 40	Not less than 89,600	12 per cent. in 2 in.	One rail tested from each 100 tons. On failure of first test, a test from another rail from same blow is allowed; if this latter test fails, all the rails of the blow are rejected
60 to 100	38 to 45	85,120–100,800	15 per cent. in 3 in. or 2 in.	Same as above.
20 to 100	40 to 48	89,600–107,520	15 per cent. in 3 in. or 2 in.	One rail tested from each 100 tons. On failure of first test, a test from two other rails from same blow is allowed; if two of the three tests fail, all the rails of the blow are rejected.
65 to 100	38 to 42	85,120–94,080	12 per cent. in 2 in.	Tests made on one or more rails from each 100 tons.
55 to 88	35.3 to 41.3	78,227–92,450	14 per cent. in 7.87 in.	One half of 1 per cent. of rails are tested. If the tested rail fails, all rails of the blow are rejected.

In compiling Table V., which shows at a glance the various inconsistencies above referred to, only the range in carbon was included, since in the specifications quoted, the manganese limits were all between 0.70 and 1.00 per cent., and the required silicon 0.10 per cent., or under.

Tensile tests should be entirely eliminated for the reasons already given; but if the engineer will not waive them, they should at least be consistent with the chemistry required, and which is furnished with every heat,—especially when the failure to meet the specified tensile strength and elongation is in itself a cause for the rejection of all the rails of the blow thus tested.

(*f*) *Specified Tensile Strength is Often Inconsistent with Specified Elongation.*—A study of Table V. shows, in one case, that for rail-steel of 47.6 tons (106,673 lb.) minimum tensile strength, the required elongation is 11 per cent. in 7.87 in.; in two cases

TABLE V.—*Comparison of the Chemical and Tensile Requirements of Some Foreign Rail-Specifications.*

Carbon Specified.	Minimum Requirements.			
	Tensile Strength per Sq. In.		Elongation	Reduction of Area.
Per Cent.	Tons.	Pounds.	Per Cent.	Per Cent.
0.30–0.35	35	78,400	20 per cent. in 2 in.	....
0.35–0.42	38	85,120	12 per cent. in 2 in.	....
0.35–0.45	38	85,120	20 per cent. in 2 in.	....
0.35–0.45	40	89,600	20 per cent.	....
0.35–0.50	37	82,880	20 per cent. in 2 in.	25
0.35–0.50	38	85,120	15 per cent. in 3 in. or 2 in.	....
0.35–0.50	38	85,120	15 per cent.	....
0.35–0.50	40	89,600	15 per cent. in 3 in. or 2 in.	....
0.35 Min.	41.3	92,450	18 per cent.	....
0.40–0.50	37	82,880	15 per cent. in 3 in.	....
0.40–0.50	40	89,600	20 per cent. in 2 in.	....
0.40–0.50	40	89,600	20 per cent. in 2 in.	....
0.40–0.50	40	89,600	20 per cent.	....
0.40–0.50	43	96,320	12 per cent. in 10 in.	20
0.40–0.50	43	96,320	12 per cent. in 10 in.	35
0.40–0.50	47.6	106,673	11 per cent. in 7.87 in.	20 to 30
0.40–0.50	47.6	106,673	8 per cent. in 7.87 in.	15 to 20
0.40–0.55	40	89,600	12 per cent. in 2 in.	....
0.40 Min.	35.3	78,227	14 per cent. in 7.87 in.	....
0.45–0.50	30	67,200	20 per cent. in 8 in.	....

where the minimum tensile is 43 tons (96,320 lb.) the steel must show 12 per cent. stretch in 10 in. With the shorter-gauged length of 2 in., a number of specifications require 20 per cent. elongation with a minimum tensile of from 28 to 40 tons (85,120 to 89,600 lb.). All these percentages of elongation are too high, especially as in one case the allowable range in carbon is from 0.40 to 0.50 per cent.

(*g*) *Amount of Tensile Testing Required is Excessive.*—In some foreign rail-specifications, the important matter of the number of tensile tests required is left in doubt by such expressions as "the inspector will occasionally test the rails for tensile strength," or "will test the rails for tensile strength from time to time." Usually, however, the number of tensile tests is definitely specified, and as the requirements are based on English practice, they form a great hardship when applied to American mills, and if literally carried out would hold back the output by causing a serious congestion in loading.

The American mills most frequently rolling rails for export produce from 1,800 to 2,200 tons of finished rails every 24 hr. Assuming 2,000 tons, and that an 80-lb. 33-ft. rail is being

rolled, this tonnage is equivalent to 5,100 rails per 24 hr. On this basis, the number of tensile tests required from American mills per day, by various foreign rail-specifications, would be as follows:

TABLE VI.—*Number of Tensile Tests in Foreign Rail-Specifications.*

Requirements of Foreign Rail-Specifications.	American Output Equivalent.
"One from each 100 tons."	20 tensile tests per day.
"One from every 250 rails."	20 tensile tests per day.
"Three every 250 rails."	60 tensile tests per day.
"One per cent. of number of rails rolled."	51 tensile tests per day.
"Not over half per cent. of number of rails rolled."	Not over 26 tensile tests per day.
"Not over 2 per cent. of day's rolling."	Not over 102 tensile tests per day.
"One from every fifth cast."	20 to 40 tensile tests per day.

But even Table VI. does not represent the amount of tensile testing required by foreign rail-specifications.

Specifications calling for a test from "one rail in every 100 tons," also stipulate that the engineer can require the manufacturer to prepare extra test-specimens, for independent test, to the extent of "two for every 200 tons of product," equivalent to 20 additional test-specimens per day.

Another specification requiring that "one or more rails from each 100 tons" be regularly tested for tensile strength, also demands that tensile specimens must be prepared and tested "from every rail which in the drop-test fails under two blows to bend through an angle of 70 degrees."

Another specification, requiring that a tensile test be cut "from one finished rail, to the extent of one per cent. of each day's rolling" (namely, 51 tests per day), also demands, "in addition to the tests made on one per cent. of the rails, a tensile test is to be made from crop-ends representing each of the remaining charges."

(*h*) *Conclusions.*—It has been said that, with specified carbon, manganese, silicon and phosphorus, and also a drop-test, it is unreasonable to demand tensile tests as well; that the tensile strength, as determined in rails, has not been proved to bear any relation to the wear of the rails in service; that it is affected by the section; that, as specified, it is often inconsistent

with the required chemistry and elongation; and that the number of tests demanded is very excessive. It is worthy of note that the three British standard rail-specifications, while specifying tensile tests, leave the enforcement of this requirement optional with the engineer.

For the reasons above given, and also because this test is often not actually required to the extent specified, it has been entirely omitted in the proposed standard rail-specification appended to this paper.

3. *Dead-Weight Test.*

(*a*) *American Specifications.*—None of the American rail-specifications include a static test. This test, like the tensile test just discussed, is, in American practice, also considered as an unnecessary check on the product, when chemical composition and a drop-test are specified.

(*b*) *Foreign Specifications.*—This transverse test under static load is still found in many foreign rail-specifications. It has been omitted in the three recent British standard rail-specifications, and is also not now included in the rail-specifications of a number of noted British engineers.

The test may be divided into two classes:

1. The test confined to subjecting the rail, laid on roller-supports a specified distance apart, to a specified load for a stated time, said load (from 12 to 28 gross tons) being within the elastic limit of the steel. After this load has been applied for a given length of time (5, 15 or 30 min.), the deflection is measured, and it is specified that this deflection shall not exceed a certain amount (usually between the limits of from $\frac{3}{16}$ to $\frac{1}{2}$ in.), and that, after the removal of the load, no permanent set shall have taken place.

2. A continuance of the above test, consisting, after the deflection due to the first application of the load has been measured, in an increase of the load, by regular increments, up to usually twice the original load; the rail must have then deflected between 1 and 2 in., and, on removal of the load, the permanent set must not be more than $1\frac{3}{8}$ in.

Studying foreign rail-specifications with reference to the first method, where the specified load is below the elastic limit, the fact that the deflection is largely a factor of the section seems,

in some cases, not to have been given due consideration. For example, while a load of 12 tons with 3 ft. between supports is specified for "T" rails weighing 43, 45, 46 and 50 lb. per yd., yet this same load is specified for 73- and 85-pound "T" rails, the only difference being 5 ft. between supports. In three specifications the specified loads for 60-lb. "T" rails are 15, 16 and 21 tons, with 3 ft. 6 in. between supports in each case. With 3 ft. between supports in both cases, the load for a 70-lb. "T" rail is in one case 22 tons, whereas in the other only a 23-ton load is required for a 90-lb. bull-head rail. In one case, with an 80-lb. "T" rail the required load is 24 tons with 4 ft. between supports, whereas with 3 ft. between supports, a 90-lb. bull-head rail is tested with a 23-ton load.

In two specifications, with the same chemistry, the same distance between supports and the same maximum deflection of $\frac{3}{8}$ in., the specified load for a 60-lb. "T" rail 4 in. in height is 16 tons, as compared with a 28-ton load for a 75-lb. bull-head rail 5 in. in height. There is another criticism that is pertinent—namely, that after requiring the manufacturer to put a piece of rail under a load below its elastic limit, there is often no mention as to the amount of deflection required; the length of time which the rail shall remain under test is also sometimes omitted.

The chief objection to the second class of dead-weight tests, where the loading is gradually brought up to double the initial load, is the length of time occupied in properly making the test.

(c) *Number of Tests Specified.*—The inconsistencies above noted in the dead-weight test, and the doubt, owing to the frequent omission of important details, as to exactly how the inspector will require the test to be made, make this requirement an uncertain factor in the cost of manufacture, and an important item in the cost, should the inspector carry out the number of tests actually specified, particularly since, in the majority of cases, it is specified that a failure to meet the requirements of the dead-weight test results, in itself, in the rejection of the blow represented by the rail tested. It is often specified that the test shall be made on a full-length rail, or a piece from 10 to 12 ft. long; in this case, the test cannot be obtained until after straightening, which leaves but little time for the actual

test, before the rails of the cast are drilled and ready to load. As to the number of dead-weight tests specified, as an actual check on the output, the same remarks as made on pp. 596–7 apply to the impracticability of such clauses as "one rail from every blow;" "one rail out of every 100;" "one rail out of every 500;" "one per cent. of the number of rails rolled per day;" "at least three rails out of every 250."

It can be positively affirmed that it is utterly impossible, with the output of American mills, to have a dead-weight test applied as a regular check on the stiffness or hardness of a percentage proportion of the output. If it is not so applied, it should not be included in the tests which, on failure, cause rejection. If it is not classed with tests causing rejection, it seems useless to make it at all, although, of course, the manufacturer should have no objection to making a few tests on each day's output to show that the rails will stand a certain specified weight without permanent set; but continued tests to be within certain deflection-limits should be waived.

In the specifications proposed as a standard and appended to this paper the dead-weight test has been omitted.

4. *Bending-Test.*

(*a*) *American Specifications.*—One or two American rail-specifications include a bending-test on bars rolled or forged from small test-ingots cast while teeming the heat. In practice, this test is not required.

(*b*) *Foreign Specifications.*—In some foreign rail-specifications certain bending-tests are given sufficient prominence to be a ground for rejection of the blow of steel if failure occurs, and they must, therefore, be duly considered in a critical review of foreign requirements, made with reference to establishing a standard rail-specification, including all necessary checks on the product, and under which satisfactory rails for export can be furnished from American mills.

In some cases, it is suggested that a small ingot be cast from metal taken by a hand-ladle from the stream during the operation of filling the first ingot-mold and similarly from the last mold when tapping the ladle into some 6 or 8 ingots, which entire operation occupies in all only from 7 to 12 min. It is required that these small ingots, 2.5 by 2.5 in., or 3 by 3 in. by 4

or 6 in. long, be rolled or forged into 0.5-in. sq. bars. A piece of this bar about 12 in. long is required to bend cold 90° without fracture. The practical value of such a test is not worth the trouble taken in making it. It originated before the present rapid and accurate methods for the determination of the carbon and manganese were perfected, and these more accurate checks, which are willingly reported on each heat as blown, should now be substituted.

In some foreign rail-specifications bending-tests are required on pieces cut from rails. In one case "one rail in every 100 tons must be bent sideways, by pressure, to a curvature of 10 ft. radius without any sign of fracture"; in another, "a short piece shall be bent by hammering cold to an angle of 95° with an internal radius of curvature of not more than 3 in. without showing fracture"; in another, the "piece of rail being placed flat must bend cold on a radius of 75 meters (246 ft.) without showing any crack"; in another, "one rail from each 100 tons must bend sideways, by pressure, to a curve of 30 ft. radius without cracking."

The British "Standards Committee" includes a full size bending-test for heavy tramway rails, but then leaves it optional with the engineer, and omits the test altogether from the standard specifications for bull-head and flat-bottomed rails.

To make these full-sized bending-tests requires heavy hydraulic presses; to make them a frequent check, which, on failure, rejects a proportion of the product, is impracticable in American practice. The specified drop-test and chemistry are a sufficient check on toughness.

For the above reasons, bending-tests on small pieces and on rails are omitted in the proposed standard specifications added to this paper.

V. Section.

1. *American Specifications.*

In 1881, A. L. Holley reported that American rail-mills had 188 different sections that were considered as standards, and that, of these, 119 patterns of 27 different weights per yd. were regularly manufactured. In 1893, a special committee of the American Society of Civil Engineers recommended for adoption standard "T" sections for 12 different weights of rail, varying

by 5 lb., from 40 to 100 lb. per yd. In February, 1906, their Rail Committee reported on statistics received from eight of the nine rail-mills in the United States, showing that, on an average, for the year ending June 30, 1905, more than 80 per cent. of their total output was rolled to what have now become popularly known as the "A. S. C. E. rail-sections."

These conditions have naturally simplified the clauses in American specifications covering the actual approval of templets, and none of the American specifications require the engineer's approval of a sample-piece of rail before rolling. It is the uniform practice in America to allow a variation of $\frac{1}{64}$ in. less and $\frac{1}{32}$ greater than the specified height, and almost all specifications contain a clause reading that "unless otherwise specified, the section of rail shall be the American Standard, recommended by the American Society of Civil Engineers, and shall conform, as accurately as possible, to the templet furnished by the railroad company, consistent with the paragraph relative to specified weight." This latter provides that the weight of the rails shall be maintained as nearly as possible after complying with the paragraph relative to allowable variation from templet; a tolerance of 0.5 per cent. in weight being allowed on an entire order.

In actual practice, these requirements have been found a sufficient check to prevent any complaint from the purchaser. Usually not more than 6,000 tons of rails are rolled without dressing rolls, and one rail every hour is weighed to maintain uniformity in the product.

2. *Foreign Specifications.*

(*a*) *Templets.*—It is specified that the manufacturer shall prepare and submit for approval, templets of both rail and fish-plates, and in some cases stud- and plug-gauges to check accuracy in drilling; later he must supply one or two full sets of templets, engraved as specified.

On export orders, American mills should comply with all reasonable requests in this regard. The duplicate templets for record are made of German silver or brass, and engraved as specified.

(*b*) *Sample Piece of Rail.*—The British standard specifications omit the requirement contained in some foreign specifications that even after the approval of templets the manufacturer must

submit short lengths of rail, prepared under ordinary conditions, with bolt-holes in position for a complete fish-joint; the rolling not to proceed until the engineer has given written notification of his approval of the sample-section of rail. It is unnecessary, after approval of templets, to require the mill to put in the rolls to furnish short sample-pieces before proceeding to fill the order, but after the rolling is regularly under way, these sample-pieces, fitted with splice-bars and bolts complete, should be willingly furnished if desired.

(c) *Allowable Variation from Templet.*—While properly insisting that the section shall be accurate to the approved templet throughout the rolling, some foreign specifications do not make provision for the unavoidable wear of the rolls. American mills agree to maintain a perfect fit of the splice-bars at all times, but require the variations in height above mentioned, which is now the standard American practice.

VI. Weight.

1. *American Specifications.*

The uniform practice as to tolerance in section is naturally followed up in American specifications by a uniform tolerance in weight, the allowable variation in all specifications being 0.5 per cent. on the entire order. Some variation from theoretical weight is unavoidable, and after limiting this variation in section and in weight to what is considered good standard practice, all American specifications then allow the rails to be accepted and paid for according to actual weights.

In practice there is no complaint from this logical method of billing rails. The mill weighs one rail every hr., which is the best and quickest method of keeping the product uniform.

2. *Foreign Specifications.*

The tolerances in weight specified by different foreign engineers are far from uniform; in some cases less and in some more leeway is given than necessary in good standard practice. Some sections are more difficult to roll than others. It is certainly unfair to bill rails on theoretical weights, or on the basis of the average of actual weights of a small percentage of each day's rolling; this necessitates considerable re-handling of finished rails, to no practical advantage.

VII. Length.

1. *American Specifications.*

Requirements and practice are uniform in recognizing as a standard either a 30- or 33-ft. rail. All specifications permit the maker to deliver shorter rails to an amount equal, if necessary, to 10 per cent. of the entire order, but these rails must be cut, in even feet, down to either 24 or 27 ft.

A variation of 0.25 in. longer or shorter than standard length is uniformly allowed.

2. *Foreign Specifications.*

There are a number of lengths defined as standard in different foreign specifications. Some specifications recognize the justice of allowing the maker to furnish some rails shorter than standard, but leave in doubt the percentage-allowance and the variations of length. Rails ordered cut to special lengths, for curves, should not be classed as short rails. If a large number of special lengths are specified, an extra charge for cold-sawing and excess of scrap should be expected. Matters such as these, having a direct bearing on the cost of manufacture, should, in justice to the bidder, be distinctly defined.

Specifications allowing only $\frac{1}{8}$ in. above or below a standard length of 40, 41 and even 45 ft., which can be met only by cold-milling the rails, seem to be an unnecessary refinement, and would act as a hardship when applied to the output of American mills, whose universal practice is hot-sawing rails to length.

VIII. Drilling.

1. *American Specifications.*

The requirements which American purchasers of rails have found sufficient to obtain a satisfactory drilling of rails by American mills are described in almost all their specifications in these words: "Circular holes for splice-bars shall be drilled in accordance with specifications of purchaser. They shall be accurate to drawing and dimensions furnished in every respect, and free from burrs."

2. *Foreign Specifications.*

The requirements for drilling are given in considerable detail in most foreign specifications. In some cases a less tolerance is allowed than $\frac{1}{32}$ in., which is an unnecessary refinement and cannot be regularly met in practice. American mills are specially equipped for the accurate drilling of circular holes, and hence object to the oval holes still occasionally specified.

IX. Finish.

1. *American Specifications.*

Under this heading, the requirements of the American specifications now in use are that the rail shall be finally straightened when cold; the ends square and clean, burrs removed and the finished rail smooth on head, straight and free from injurious defects and flaws of all kinds.

2. *Foreign Specifications.*

In foreign specifications, the operations of finishing are described more in detail, and all the defects from which the rail must be free are enumerated. The British standard specifications devote very little space to these details, but probably sufficiently cover the ground, as in practice each rail is submitted to a careful surface inspection both by the manufacturer's men and by the representative of the purchaser.

X. Branding.

1. *American Specifications.*

In American practice all branding is done on the hot rail. The name of the maker, and the month and year of manufacture, are rolled in raised letters on the web of each rail.

While the rail is still hot, the blow-number is stamped on the web at least twice, and far enough from the ends to insure that the numbering can be seen after the rail is in track.

It is customary to identify "Seconds," or No. 2 rails, and short rails and those of special length by painting the ends. The standard practice in America is to use white paint for No. 2 rails and green paint for "shorts."

2. *Foreign Specifications.*

(*a*) *Data to be Rolled in Raised Letters on Web of Rail.*—Foreign engineers specify considerable data to be rolled in relief on the web of each rail. Besides the maker's name, initials or other recognized mark and the month and year of rolling, the initials of the railroad are generally desired, and sometimes the weight of the rail per yard.

The British standard specifications also require a special "Brand" and the number of the "B. S." section to be rolled on the web, which shall show that the rail is of British standard section and made under the conditions of their specification. They also specify that the process by which the rails have been made shall be added, and recommend the following abbreviations:

S. A. Siemens-Martin Acid.	B. A. Bessemer Acid.
S. B. Siemens-Martin Basic.	B. B. Bessemer Basic.

The addition of the word "steel" required by some engineers seems superfluous, as iron rails now in track can be recognized on inspection, and practically none are now rolled. In the United States the output of iron rails for 1905 was 318 tons, and in Great Britain practically *nil.*

The requirement that, in addition, a distinct mark shall be rolled on the web, said mark to be chipped off all rejected rails, is certainly unnecessary, since it is perfectly safe to rely on the integrity of the rail-makers not to attempt to ship rails once rejected by the inspector.

(*b*) *Marking of the Blow-Number on Each Rail.*—The majority of foreign specifications require one or both ends of each rail to be stamped with the number of the blow. This stamping, of course, must be done cold, and, at best, will not be very legible; furthermore, the number cannot be seen when the rail is in track. The foreign engineer will gladly modify this to suit the American practice of mechanically stamping the blow-number, two or three times, on the web of the hot rail.

(*c*) *Additional Marking and Painting of Rails.*—Foreign specifications usually stipulate that the inspector shall brand each rail with his official stamp. This the American mills should allow, but in view of their output of 5,100 rails per 24 hr., the

inspector should arrange to do this branding without the necessity of re-handling all the rails for this special purpose.

It is fair to specify that rejected rails shall be at once so marked as to be subsequently identified during rolling, but they should not be stenciled with the word "rejected," or otherwise branded so as to render them unsalable to other parties, should the maker so desire. The stamping of the length on each rail should be waived, as it retards loading.

The identification, by paint of different colors, of short and special-length rails, is usual practice, but the covering of the entire rail with anti-corrosive paint, or brushing it all over with boiling linseed-oil, are requirements which so seriously retard the automatic loading-facilities of American mills that they desire to have this requirement waived, even if it can be classed as an extra.

XI. Inspection.

1. *American Specifications.*

The defining of the relations between the maker and the inspector occupies very little space in American rail-specifications.

Representatives of the purchaser are given free entry to the works and all the help they ask for; the maker gives due notice when rolling will begin, and he knows that any disputes arising will be at once settled on their merits, that all tests and inspection will be made at the mills, and that there will be no serious delays in his shipment of the accepted rails.

2. *Foreign Specifications.*

The requirements of inspection found in different parts of foreign specifications are given in such detail that, by the following subdivisions, they can be more conveniently discussed:

(*a*) *Prior Notice of Rolling.*—This requirement is always complied with by the rail maker, but in return he should be given permission to proceed at the time designated, if the inspector is not present.

(*b*) *Free Access to the Works.*—This privilege is always accorded, but foreign engineers are unnecessarily explicit in insuring that their representative shall be given all the tools, testing-appliances, gauges, templets, etc., required, and that all necessary labor and assistance shall be furnished to enable the

inspector to fulfill his duties. American mills are noted for their willingness to furnish, free of cost, every facility necessary for inspection; but in return, they have a right to ask that the inspection shall be intelligently carried out, that decisions shall be promptly made, and that they shall be assured that no serious delays in manufacturing-operations shall occur during rolling, due to the absence of the inspector or his inability to follow up to his satisfaction all the requirements of the specification.

(c) *Cost of Inspection, Analyses and Tests to be Borne by the Manufacturer.*—The requirement that the maker shall pay all costs connected with whatever chemical analyses or physical tests by independent laboratories the inspector may demand, leaves an item of the cost of manufacture in considerable doubt. American mills object to any testing by independent laboratories and the rejection on their results, without appeal, in part because the laboratories are so far distant, and also because the samples sent may not be representative, and in some cases the results may be inaccurate. When checks by outside parties are demanded the purchaser should pay the costs.

(d) *Rails after Inspection by Manufacturer to be Sorted into Lots Prior to Inspection by the Engineer.*—The provisions of foreign specifications, requiring that after the mill-inspection the finished rails shall be "sorted into lots," usually of uniform length, before examination by the representative of the purchaser, cannot be lived up to in American mills. Ample opportunity for a thorough inspection of the finished rail is given, but the re-handling necessary to comply with the above provision is impracticable and should be waived.

(e) *Disposition of Rejected Rails.*—The requirements that "rejected rails shall be stacked and kept apart until completion of contract" and that "no rejected rails shall be sold or consigned to scrap until the completion of the contract" are entirely unreasonable, especially when the contract is, for any reason, not completed at one continuous rolling.

(f) *No Appeal from Engineer's Decision.*—The foreign engineer's "written certificate" is necessary before any portion of the rails can be considered as a delivery on the contract; but even this certificate does not relieve the contractor from responsibility for replacement, and it is not obligatory on the pur-

chaser to allow any extension of time in which to deliver substituted material. Some specifications delegate "full power to the inspector to reject, without appeal," all or any part of the rolling which, in his opinion, does not comply with the specification, or which is in any way defective; others state that in case of dispute, "the engineer's decision shall be final and binding"; another specification, after stating that the material shall be made in accordance with its requirements, also prescribes that the manufacture shall be "under the control, direction and supervision of the inspector, whose instructions on all points relating to the nature and quality of materials used and workmanship executed are to be received and acted on by the makers."

While fully recognizing the purchaser's rights, as well as those assumed by his engineer, no manufacturer should enter into a contract if he felt that such clauses as the above were to be interpreted literally. The British standard specifications contain no such requirements.

(*g*) *Final Acceptance of Rails at Port of Delivery.*—To require that the final acceptance of rails shall be at the port of delivery is not a businesslike proposition; no maker of rails should be required to guarantee "to deliver said rails in perfectly good order and condition at the places of destination."

(*h*) *Rejections may be Made After Delivery.*—Some foreign engineers specify that, even after the rails have been passed at the mills and paid for, the maker must remove and replace any rails found defective by an inspection and testing instituted after delivery; it is also sometimes specified that the rails must be guaranteed for a stated period, two, five, or, in one case, ten years, after having been put in track.

While it may be assumed that American makers stand ready to adjust any complaint proving that they have furnished defective material, they would naturally be unwilling to guarantee service for a certain period, because the distant port at which the rails may be delivered makes an investigation difficult, and it is unreasonable to expect them to replace rails, unless the failure in service is unquestionably the fault of the maker, a question that can be fairly determined only by representatives of both sides.

A recent committee of railroad engineers was excused from

drawing conclusions on statistics received as to the life of rails, on making the following concise and truthful report: "The life of rails is affected by many conditions, such as alignment, profile, density of traffic and speed of trains."

XII. Seconds, or No. 2 Rails.

1. *American Specifications.*

The inspection of the "finish" of a rail is rigid; as already noted under this heading, the rail must be free from injurious defects and flaws of all kinds. As it is impossible to make every rail perfect, the definition of what shall constitute "seconds," or "No. 2 rails," has been incorporated in American specifications, and it has become usual practice for purchasers to accept these rails for use in the main track at stations, in sidings and in yards, at a less price and in amount up to 5 per cent., and in the very heavy sections 10 per cent., in addition to the tonnage ordered. These rails are carefully identified by white paint on each end.

The standard definition of "seconds," or "No. 2 rails," found in American specifications is as follows: "Rails which possess any injurious physical defects, or for any other cause, are not suitable for first quality, or No. 1 rails, shall be considered No. 2 rails."

2. *Foreign Specifications.*

No reference to "seconds," or "No. 2 rails," is found in foreign specifications. American railroads have found it to their advantage to use these cheaper rails for the purposes mentioned, and a clause covering their inspection has therefore been inserted in the proposed standard specification appended to this paper.

XIII. Proposed Standard Specification to Govern the Manufacture, in American Mills, of Steel Rails for Export.

In the following specification, proposed as a standard to govern the rolling of American rails for export, an effort has been made to avoid ambiguity, to bring related requirements together under one heading, to make them definite and concise, but to leave the specification flexible enough to allow an intel-

ligent inspector to use his discretion in doubtful cases without jeopardizing the duty he owes to the purchaser; it provides that his decision shall be prompt, so as not to hamper unnecessarily the manufacturing-operations when the maker, in good faith, has agreed, in signing the contract, strictly to live up to the conditions required.

1. *Process of Manufacture.*

(*a*) The steel shall be of the best quality and made by the Bessemer or the Siemens-Martin process.

(*b*) The materials used and the entire process of manufacture and testing shall be in strict accordance with the best standard current practice, and special care shall be taken to conform to the following instructions.

(*c*) No cracked or badly-patched molds shall be used, and the ingots shall be kept in a vertical position in the pit heating-furnaces until ready to be rolled, or until the metal in the interior has had time to solidify.

(*d*) No "bled" ingots shall be used, and no ingots from "chilled" heats rolled into first-quality rails. A "bled" ingot is one from the center of which liquid steel has escaped. A "chilled" heat is one which, because of the cooling of the steel, has to be either pricked or poured over the top of the ladle.

(*e*) Sufficient material shall be discarded or "cropped" from the top of all ingots, to insure sound rails.

(*f*) Under no circumstances shall an ingot or rail-bloom be heated so high that the cinder on it runs, when being drawn from the soaking-pit, or heating-furnace. The ingots or blooms must be evenly heated throughout their length, drawn at a uniform temperature, and a uniform finishing-temperature also maintained.

2. *Chemical Composition.*

Rails of the various weights per yard specified below shall conform to the limits in chemical composition shown in Table VII.

3. *Chemical Analyses.*

The manufacturer shall make and furnish to the representative of the engineer (or of the purchaser), before the rails rolled on each turn are ready for shipment, determinations of carbon

TABLE VII.—*Chemical Composition.*

	50 to 60 Lb.	Over 60 to 70 Lb.	Over 70 to 80 Lb.	Over 80 to 90 Lb.	Over 90 to 100 Lb.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Carbon......	0.30–0.50	0.35–0.45	0.35–0.45	0.40–0.50	0.40–0.50
Manganese..	0.75–0.95	0.75–0.95	0.75–0.95	0.80–1.00	0.80–1.00
Silicon......	Not over 0.10	Not over 0.10	Not over 0.10	Not over 0.10	Not over 0.10
Phosphorus	Not over 0.10	Not over 0.10	Not over 0.10	Not over 0.10	Not over 0.10
Sulphur.....	Not over 0.07	Not over 0.07	Not over 0.07	Not over 0.07	Not over 0.07

and manganese on each heat of steel; determinations of phosphorus on six heats every 12 hr., and determinations of silicon and sulphur on a sample representing each 12 hr. rolling; all said analyses to be made on drillings from a test-ingot cast when teeming each heat.

The results of these analyses shall govern; but prior to and during the rolling of the rails, the inspector shall be given all reasonable facilities to satisfy himself that the maker's method of sampling each heat is in accordance with the best standard practice, and that the various constituents of the rail-steel will be accurately determined. If so desired, but prior to rolling, the inspector may submit to the manufacturer and to an independent chemist of repute, a properly-prepared sample of rail-steel for joint analysis, with a view to proving the accuracy of the analyses which will be furnished daily, from the mill-laboratory, when rolling begins; the expense of this joint-analysis to be equally divided between the contractor and the purchaser.

If, during rolling, the accuracy of any report from the steel-works laboratory on a heat of steel is promptly questioned, and not immediately settled to the inspector's satisfaction by a check-analysis at the mill, the inspector shall be furnished an identified sample of drillings from the test-ingot, the analysis of which, by an independent chemist of repute, must be furnished and the dispute settled before the rails from said heat are ready for shipment. The purchaser shall pay the cost of any and all such check-analyses.

If requested, the manufacturer shall furnish, on an average sample of the tonnage rolled, a determination of arsenic, copper, or other constituents, incidentally present in determinable quantities.

4. *Impact-Test.*

From either end of a rail, from every alternate heat of steel, shall be cut, at the hot saws, a piece from 4 to 6 ft. long, which shall be distinctly stamped with the heat-number, as directed by the representative of the engineer (or of the purchaser), and the piece set aside on skids to cool. As soon as cool, it shall be placed, head upwards, on the supports of the standard American rail drop-testing machine, described below, and the various sections must withstand, without fracture, one blow of the 2,000-lb. tup, from the height specified in Table VIII. The report of the drop-test shall include a record of the atmospheric temperature at the time the tests are made, and, if necessary, due allowance shall be made for rails tested at or below freezing-temperature.

TABLE VIII.—*Falling-Weight Test.*

Weight of Rail.	Height of Drop.	Range in Carbon.
Pounds Per Yard.	Feet.	Per Cent.
50, up to and including 60..........	16	0.30–0.40
More than 60, up to and including 70.........	17	0.35–0.45
More than 70, up to and including 80.........	18	0.35–0.45
More than 80, up to and including 90.........	19	0.40–0.50
More than 90, up to and including 100.........	20	0.40–0.50

With regard to a provision for re-test, in case the pieces of rail selected to represent the heat fail under test, two other rails will be selected and similar lengths cut therefrom, or crop-ends of sufficient length already cut from two other rails of the same heat may be used. If either of these pieces or crop-ends fail, all the rails of the heat which they represent will be rejected, but if both these latter tests meet the requirements, all the rails of the heat will be accepted.

In case the rails from two successive tested heats are rejected for failure to meet the requirements of the drop-test, the intermediate heat will be tested as above described.

The acceptance or rejection of all the rails from any heat will depend upon the result of the drop-test thereof.

5. *Drop-Test Machine.*

Prior to making any impact tests, the inspector shall be given opportunity to satisfy himself that the drop-testing machine, in

its essential features, complies with the following requirements: the weight of the tup shall be 2,000 lb.; the radius of its striking-face 5 in.; the weight of the anvil-block shall be amply sufficient to insure rigidity, and, moreover, it shall rest on a solid masonry foundation. The iron or steel supports for the rails shall be rounded to a radius of 5 in., set 3 ft. apart between centers, and shall be firmly secured to the anvil-block. The tup shall be so arranged that it can conveniently be dropped from any specified height.

6. *Section.*

Before the general manufacture of the rails is commenced, the manufacturer shall, if required by the engineer (or by the purchaser), supply two sets of templets, internal and external, made of approved material. These templets, engraved as specified in the contract, shall be submitted to the engineer (or to the purchaser) for his approval, and at the commencement of rolling the engineer shall have a competent person present to approve of the section.

The rails shall be of uniform section throughout, and shall conform, as accurately as possible, to the approved templet, consistent with paragraph No. 7 relative to specified weight. To allow for the unavoidable wear of the rolls, a variation in height of $\frac{1}{64}$ in. less and $\frac{1}{32}$ in. greater than the specified height will be permitted. A perfect fit of the fish-plates, however, shall be maintained at all times.

7. *Weight.*

The weight of the rails shall be maintained as nearly as possible, after complying with paragraph No. 6, to that specified in the contract. A variation of 0.5 per cent. for an entire order will be allowed. The manufacturer shall weigh one rail each hour during the entire rolling.

Rails shall be accepted and paid for according to actual weight.

8. *Length.*

The standard length of rails shall be 33 ft. Ten per cent. of the entire order, if made, will be accepted in shorter lengths, varying by even feet down to 27 ft. A variation of 0.25 in. longer or shorter than the standard length will be allowed.

This allowable variation shall be $\frac{3}{8}$ in. longer or shorter for rails over standard length.

When rails of special lengths, slightly shorter than 33 ft., are ordered for curves, they shall not be counted in the allowable percentage of short rails.

9. *Drilling.*

Circular holes for fish-bolts shall be drilled through the web, from the solid, at each end of the rails and in strict accordance with the specification. They shall conform accurately to the drawing and dimensions furnished in every respect, shall be clean and square with the web, and shall be left without burrs on either side.

Should any of the holes vary from the correct size or position more than $\frac{1}{32}$ in., the rails in question will be liable to rejection.

10. *Finish.*

Rails shall be straight in line and surface when finished; the straightening being carefully done while cold; smooth on head, sawed square at ends (variation to be not more than $\frac{1}{32}$ in.), and, prior to shipment, shall have the burr occasioned by the saw-cutting carefully chipped and filed off, particularly under the head and on top of the flange; the ends must be clean. The rails are to be free from injurious defects and flaws of all kinds.

11. *Branding.*

The maker's name, initials or other recognized mark, the month and year of manufacture and the initials of the railroad, shall be rolled in raised letters on the side of the web. If specially desired, the weight of the rail per yard will be added.

The heat-number shall be plainly stamped on the web of each rail while hot, in at least two places and at a sufficient distance from the ends, so that it will not be subsequently covered by the fish-plate.

All rails definitely rejected shall be at once marked in such a distinctive manner as will enable the inspector to readily identify them, but not so as to render them unsalable to other parties.

Both ends of all "seconds," or No. 2 rails, to be painted white. Both ends of all short lengths first quality, or No. 1,

rails to be painted green. Special-length rails for curves to be painted red.

12. *Inspection.*

The manufacturer shall give the engineer (or the purchaser), or his inspector, if so instructed, a reasonable notice, in writing, before rolling shall be begun, and similar written notices in advance of each resumption of rolling, in case the order is not filled at one continuous rolling. Should the maker fail to give said notice, all rails rolled in the absence of the duly authorized representative may be rejected as part of the contract. The party thus notified shall, in turn, give written notice to the manufacturer of his intention to be present, or permission to proceed at the time designated by the maker.

Authorized representatives shall have free access to the works of the manufacturer at all times when the contract is being filled, and shall have, free of cost, all reasonable facilities afforded by the maker to satisfy them that the finished rails are furnished in accordance with the terms of these specifications; the inspection shall, therefore, be conducted so as to cause no serious delays in the processes of manufacture.

Rails will be passed upon individually or by heats, according to the character of the requirements specified. Rejected rails will become at once the property of the maker.

All tests and inspection shall be made at the place of manufacture, and the engineer, or his representative at the mill, shall be empowered to give the necessary written certificates of acceptance to the manufacturer, in such a manner as not to cause delays in the shipment of inspected rails.

13. *"Seconds," or No. 2 Rails.*

Rails which possess any injurious defects, or which, from any other cause, are not suitable for first quality, or No. 1, rails, shall be considered as "seconds," or No. 2 rails.

They shall not have flaws in their heads of more than 0.25 in., or in the flange of more than 0.5 in. in depth, and, in the judgment of the inspector, these shall not be so numerous or of such a character as to render them unfit for recognized No. 2 rail uses, such as in the main track at stations, in sidings and in yards.

The ends of No. 2 rails shall be painted white, and they shall

have two prick-punch marks on the side of the web near the blow-number brand, and placed so as not to be covered by the fish-plates.

No. 2 rails will be accepted up to 5 per cent. of the whole order.

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The Lime-Roasting of Galena.

BY W. R. INGALLS, NEW YORK, N. Y.

(London Meeting, July, 1906.)

DURING the last two years, and especially during the last six months, a number of important articles upon the new methods for the desulphurization of galena have been published in the technical periodicals, particularly in the *Engineering and Mining Journal* and in *Metallurgie*. I proposed for these methods the type-name of "lime-roasting" of galena as a convenient metallurgical classification,[1] and this term has found some acceptance. The articles referred to have shown the great practical importance of these new processes, and the general recognition of their metallurgical and commercial value which has already been accorded to them. It is my present purpose to review broadly the changes developed by them in the metallurgy of lead, in which connection it is necessary to refer briefly to the previous state of the art.

[1] *Engineering and Mining Journal*, vol. lxxx., p. 402 (1905.)

The elimination of the sulphur-content of galena has been always the most troublesome part of the smelting-process, being both costly in the operation and wasteful of silver and lead. Previous to the introduction of the Huntington-Heberlein process at Pertusola, Italy, it was effected by a variety of methods. In the treatment of non-argentiferous galena concentrate, the smelting was done by the roast-reduction method (roasting in reverberatory furnace and smelting in blast-furnace); the roast-reaction method, applied in reverberatory furnaces; and the roast-reaction method, applied in Scotch hearths.[2] Precipitation-smelting, simple, had practically gone out of use, although its reactions enter into the modern blast-furnace practice, as do also those of the roast-reaction method.

In the treatment of argentiferous lead-ores, a combination of the roast-reduction, roast-reaction and precipitation-methods had been developed. Ores low in lead were still roasted, chiefly in hand-worked reverberatories (the mechanical furnaces not having been proved well adapted to lead-bearing ores), while the high loss of lead and silver in sinter- or slag-roasting of rich galenas had caused those processes to be abandoned, and such ores were charged raw into the blast-furnace, the part of their sulphur which escaped oxidation therein reappearing in the form of matte. In the roast-reduction smelting of galena alone, however, there was no way of avoiding the roasting of the whole, or at least a very large percentage of the ore, and in this roasting the ore had necessarily to be slagged or sintered in order to eliminate the sulphur to a satisfactory extent. This is exemplified in the treatment of the galena concentrate of southeastern Missouri at the present time.

Until the two new Scotch-hearth plants at Alton and Collinsville, Ill., were put in operation, the three processes of smelting the southeastern Missouri galena were about on an equal footing. Their results per ton of ore containing 65 per cent. of lead were approximately as follows. (Percentages of lead in Missouri practice are based on the wet-assay; among the silver-lead smelters of the West the fire-assay is still generally employed.)

[2] This term is inexact, because the hearths employed in the United States are not strictly "Scotch hearths," but they are commonly known as such, wherefore my use of the term.

Method.	Cost.	Extraction. Per Cent.
Reverberatory, ...	\$6.50–\$7.00	90–92
Scotch-hearth, ...	\$5.75–\$6.50	87–88
Roast-reduction, ...	\$6.00–\$7.00	90–92

The new works employ the Scotch-hearth process, with bag-houses for the recovery of the fume, which previously was the weak point of this method of smelting. This improvement did not originate at either Alton or Collinsville, having been previously in use at the works of the Missouri Smelting Company at Cheltenham, St. Louis, but the idea originated from the practice of the Picher Lead Company, of Joplin, Mo. This improvement led to a large increase in the recovery of lead, so that the entire extraction is now approximately 98 per cent. of the content of the ore, while, on the other hand, the cost of smelting per ton of ore has been reduced through the increased size of these plants and the introduction of improved means for handling ore and material. The practice of these works represents the highest efficiency yet obtained in this country in the smelting of high-grade galena-concentrate, and probably it cannot be equaled even by the Huntington-Heberlein and similar processes. The Scotch-hearth and bag-house process is therefore the one of the older methods of smelting which will survive.

In the other methods of smelting, a large proportion of the cost is involved in the roasting of the ore, which amounts in hand-worked reverberatory furnaces to from \$2 to \$2.50 per ton. Also, the larger proportion of the loss of metal is suffered in the roasting of the ore, this amounting to from 6 to 8 per cent. of the metal content of such ore as is roasted. The loss of lead in the combined process of treatment depends upon the details of the process. The chief advantage of lime-roasting in the treatment of this class of ore is in the higher extraction of metal which it affords. This should rise to 98 per cent. That figure, indeed, has been surpassed in operations on a large scale, extending over a considerable period.

In the treatment of the argentiferous ores of the West, different conditions enter into the consideration. In the working of those ores, the present practice is to roast only those which are low in lead, and charge raw into the blast-furnace the rich galenas. The cost of roasting is from \$2 to \$2.50 per ton; the

cost of smelting is about \$2.50 per ton. On the average about 0.4 ton of ore has to be roasted for every ton that is smelted. The cost of roasting and smelting is therefore about \$3.50 per ton. In good practice the recovery of silver is about 98 per cent. and of lead about 95 per cent., reckoned on the fire-assay.

In the treatment of these ores, the lime-roasting process offers several advantages. It may be performed at less than the cost of ordinary roasting. (This refers especially to the Savelsberg process.) The loss of silver and lead during the roasting is reduced to insignificant proportion. The sulphide-fines which must be charged raw into the blast-furnace are eliminated, inasmuch as they can be efficiently desulphurized in the lime-roasting pots without significant loss; all the ore to be smelted in the blast-furnace, therefore, can be delivered to it in lump form, whereby the speed of the blast-furnace is increased and the wind-pressure required is decreased. Finally, the percentage of sulphur in the charge is reduced, producing a lower matte-fall, or no matte-fall whatever, with consequent saving in expense of retreatment. In the case of a new plant, the first cost of construction and the ground-space occupied are materially reduced. Before discussing more fully the extent and nature of these savings, it is advisable to point out the differences among the three processes of lime-roasting that have already come into practical use.

In the Huntington-Heberlein process, the ore is mixed with suitable proportions of limestone or silica (or quartzose ore), and is then partially roasted, say, to reduction of the sulphur to one-half. The roasting is done at a comparatively low temperature, and the loss of metals is consequently small. The roasted ore is dampened and allowed to cool. It is then charged into a hemispherical cast-iron pot, with a movable hood which covers the top and leads away the gases. There is a perforated grate in the bottom of the pot, on which the ore rests, and air is introduced through a pipe entering the bottom of the pot, under the grate. A small quantity of red-hot calcines from the roasting-furnaces is thrown on the grate to start the reaction; a layer of cold, semi-roasted ore is put upon it, the air-blast is turned on and reaction begins, which manifests itself by the copious evolution of sulphur-fumes. These consist chiefly of sulphur dioxide, but they contain more or less trioxide, which

is evident from the solution of copperas that trickles from the hoods and iron smoke-pipes, wherein the moisture condenses. As the reaction progresses, and the heat creeps up, more ore is introduced, layer by layer, until the pot is full. Care is taken by the operator to compel the air to pass evenly and gently through the charge, wherefore he is watchful to close blow-holes which develop in it. At the end of the operation, which may last from 4 to 18 hr., the ore becomes red hot at the top. The hood is then pushed up, and the pot is turned on its trunnions, by means of a hand-operated wheel and worm-gear, until the charge slides out, which it does as a solid, semi-fused cake. The pot is then turned back into position. Its design is such that the air-pipe makes automatic connection, a flanged pipe, cast with the pot, settling upon a similarly flanged pipe communicating with the main, a suitable gasket serving to make a tight joint. The pots are set at an elevation of about 12 ft. above the ground, so that when the charge slides out the drop will break it up to some extent; moreover, it is caused to fall on a wedge, or similar contrivance, to assist the breakage. After cooling, it is further broken up to furnace-size by wedging and sledging; the lumps are forked out, and the fines screened and returned to a subsequent charge for completion of their desulphurization.

The Savelsberg process differs from the Huntington-Heberlein in respect to the preliminary roasting, which, in the Savelsberg process, is omitted, the raw ore, mixed with limestone and silica, being charged directly into the converter. The Savelsberg converter is supported on a truck, instead of being fixed in position, but otherwise its design and management are quite similar to those of the Huntington-Heberlein converter. In neither case are there any patents on the converters. The patents are on the processes. In view of the litigation that has already been commenced between their respective owners, it is interesting to examine the claims.

The Huntington-Heberlein patent[3] has the following claims:

1. The herein-described method of oxidizing sulphide-ores of lead preparatory to reduction to metal, which consists in mixing with the ore to be treated an oxide of an alkaline-earth metal, such as calcium oxide, subjecting the mixture to heat in the presence of air, then reducing the temperature, and finally

[3] U. S. Patent No. 600,347, issued March 8, 1898, applied for December 9, 1896.

passing air through the mass to complete the oxidation of the lead, substantially as and for the purpose set forth.

2. The herein-described method of oxidizing sulphide-ores of lead preparatory to reduction to metal, which consists in mixing calcium oxide or other oxide of an alkaline-earth metal with the ore to be treated, subjecting the mixture, in the presence of air, to a bright-red heat (about 700° C.), then cooling down the mixture to a dull-red heat (about 500° C.), and finally forcing air through the mass until the lead-ore, reduced to an oxide, fuses, substantially as set forth.

3. The herein-described method of oxidizing lead sulphide in the preparation of the same for reduction to metal, which consists in subjecting the sulphide to a high temperature in the presence of an oxide of an alkaline-earth metal, such as calcium oxide, and oxygen, and then lowering the temperature, substantially as set forth.

Adolf Savelsberg claims:[4]

1. The herein-described process of desulphurizing lead-ores, which consists in mixing raw ore with limestone and then subjecting the mixture to the simultaneous application of heat and a current of air in sufficient proportions to substantially complete the desulphurization in one operation, substantially as described.

2. The herein-described process of desulphurizing lead-ores, which process consists in first mixing the ores with limestone, then moistening the mixture, then filling it without previous roasting into a chamber, then heating it and treating it by a current of air, as and for the purpose described.

3. The herein-described process of desulphurizing lead-ores, which consists in mixing raw ores with limestone, then filling the mixture into a chamber, then subjecting the mixture to the simultaneous application of heat and a current of air in sufficient proportions to substantially complete the desulphurization in one operation, the mixture being introduced into the chamber in partial charges introduced successively at intervals during the process, substantially as described.

4. The herein-described process of desulphurizing lead-ores, which process consists in first mixing the ores with limestone,

[4] U. S. Patent No. 755,598, issued March 22, 1904, applied for December 18, 1903.

then moistening the mixture, then filling it without previous roasting into a chamber, then heating it and treating it by a current of air, the mixture being introduced into the chamber in partial charges introduced successively at intervals during the process, as and for the purpose described.

5. The herein-described process of desulphurizing lead-ores, which process consists in first mixing the ores with sufficient limestone to keep the temperature of the mixture below the melting-point of the ore, then filling the mixture into a chamber, then heating said mixture and treating it with a current of air, as and for the purpose described.

6. The herein-described process of desulphurizing lead-ores, which process consists in first mixing the ores with sufficient limestone to mechanically separate the particles of galena sufficiently to prevent fusion, and to keep the temperature below the melting-point of the ore by the liberation of carbon dioxide, then filling the mixture into a chamber, then heating said mixture and treating it with a current of air, as and for the purpose described.

The Carmichael-Bradford process differs from the Savelsberg by the treatment of the raw ore mixed with gypsum instead of limestone, and differs from the Huntington-Heberlein both in respect to the use of gypsum and the omission of the preliminary roasting. The Carmichael-Bradford process has not been threatened with litigation, so far as I am aware. The claims of its original patent read as follows:[5]

1. The process of treating mixed sulphide-ores, which consists in mixing with said ores a sulphur compound of a metal of the alkaline earths, starting the reaction by heating the same, thereby oxidizing the sulphide and reducing the sulphur compound of the alkali metal, passing a current of air to oxidize the reduced sulphur compound of the metal of the alkalies preparatory to acting upon a new charge of sulphide-ores, substantially as and for the purpose set forth.

2. The process of treating mixed sulphide-ores, which consists in mixing calcium sulphate with said ores, starting the reaction by means of heat, thereby oxidizing the sulphide-ores, liberating sulphurous-acid gas, and converting the calcium sulphate into calcium sulphide, and oxidizing the calcium sul-

[5] A. D. Carmichael, U. S. Patent No. 705,904, July 29, 1902.

phide to sulphate preparatory to treating a fresh charge of sulphide-ores, substantially as and for the purpose set forth.

The process described[6] by W. S. Bayston, of Melbourne, appears to be identical with that of Savelsberg.

Irrespective of the validity of the Savelsberg and Carmichael-Bradford patents, and without attempting to minimize the ingenuity of their inventors and the importance of their discoveries, it must be conceded that the merit for the invention and introduction of lime-roasting of galena belongs to Thomas Huntington and Ferdinand Heberlein. The former is an American, and this is the only claim that the United States can make to a share in this great improvement in the metallurgy of lead. It is to be regretted, moreover, that of all the important lead-smelting countries of the world, America has been the most backward in adopting it.

The details of the three processes and the general results accomplished by them have been rather fully described in a series of articles recently published in the *Engineering and Mining Journal.* There has been, however, comparatively little discussion as to costs; and, unfortunately, the data available for analysis are extremely scanty, due to the secrecy with which the Huntington-Heberlein process, the most extensively exploited of the three, has been veiled. Nevertheless, I may attempt an approximate estimation of the various details, taking the Huntington-Heberlein process as the basis.

The ore, limestone and silica are crushed to pass a 4-mesh screen. This is about the size to which it would be necessary to crush as preliminary to roasting in the ordinary way, wherefore the only difference in cost is the charge for crushing the limestone and silica, which in the aggregate may amount to one-sixth of the weight of the raw sulphide, and may consequently add 2 to 2.5c. to the cost of treating a ton of ore. The mixing of ore and fluxes may be costly or cheap, according to the way of doing it. If done in a rational way it ought not to cost more than 10c. per ton of ore, and may come to less. The delivery of the ore from the mixing-house to the roasting-furnaces ought to be done entirely by mechanical means, at insignificant cost.

The Heberlein roasting-furnace, which is used in connection

[6] Australian Patent No. 2,862.

with the "H.-H." process, is simply an improvement on the old Brunton calciner—a circular furnace, with revolving hearth. The construction of this furnace, according to American designs, is excellent. The hearth is 26 ft. in diameter; it is revolved at slow speed, and requires about 1.5 h.p. A flange at the periphery of the hearth dips into sand in an annular trough, thus shutting off air from the combustion-chamber, except through the ports designed for its admittance. The mechanical construction of the furnace is workmanlike, and the mechanism under the hearth is easy of access and comfortably attended to.

A 26-ft. furnace roasts about 80,000 lb. of charge per 24 hr. In dealing with an ore containing from 20 to 22 per cent. of sulphur, the latter is reduced to about 10 or 11 per cent., the consumption of coal being about 22.5 per cent. of the weight of the charge. The hearth-efficiency is about 150 lb. per sq. ft., which, in comparison with ordinary roasting, is high. The coal-consumption, however, is not correspondingly low. Two furnaces can be managed by one man per 8-hr. shift. On the basis of 80 tons of charge ore per 24 hr., the cost of roasting should be approximately as follows:

Labor: 3 men at $2.50, . .	$ 7.50
Coal: 18 tons at $2, . .	36.00
Power, ...	3.35
Repairs, ...	3.35
Total, ...	$50.20 for 80 tons, or 63c. per ton.

In the above estimate repairs have been reckoned at the same amount as is experienced with Brückner cylinders, and the cost of power has been allowed for with fair liberality. The estimated cost of 63c. per ton is comparable with the $1.10 to $1.45 per ton, which is the result of roasting in Brückner cylinders in Colorado, reducing the ore to from 4.5 to 6 per cent. of sulphur.

The Heberlein furnace is built up to considerable elevation above the ground-level, externally somewhat resembling the Pearce turret-furnace. This serves two purposes: (1) it affords ample room under the hearth for attention to the driving mechanism; and (2) it enables the ore to be discharged by gravity into suitable hoppers, without the construction of subterranean gangways. The ore discharges continuously from the furnace,

at dull-red heat, into a brick bin, wherein it is cooled by a water-spray. Periodically, a little ore is diverted into a side-bin, in which it is kept hot for starting a subsequent charge in the converter.

The cooled ore is conveyed from the receiving-bins at the roasting-furnaces to hopper-bins above the converters. If the tramming be done by hand the cost, with labor at 25c. per hr., may be approximately 12.5c. per ton of ore, but this should be capable of considerable reduction by mechanical conveyance.

The converters are hemispherical pots of cast-iron, 9 ft. in diameter at the top and about 4 ft. in depth. They are provided with a circular, cast-iron grate, which is 0.75 in. thick and 6 ft. in diameter, and is set and secured horizontally in the pot. This grate is perforated with holes 0.75 in. in diameter, 2 in. apart, center to center, and is similar to the Wetherill grate employed in zinc oxide manufacture. The pot itself is about 2.5 in. thick at the bottom, thinning to about 1.5 in. at the rim. It is supported on trunnions, and is geared for convenient turning by hand. The blast-pipe, which enters the pot at the bottom, is 6 in. in diameter.

Two roasting-furnaces and six converters are rated nominally as a 90-ton plant. This rating, however, is considerably in excess of the actual capacity, at least on certain ores. The time required for desulphurization in the converter apparently depends a good deal upon the character of the ore. The six converters may be arranged in a single row, or in two rows of three in each. They are set so that the rim of the pot, when upright, is about 12 ft. above the ground-level. A platform gives access to the pots. One man per shift can attend to two pots. His work consists in charging them, which is done by gravity, spreading out the charge evenly in the pot, closing any blow-holes which may develop, and at the end of the operation raising the hood (which covers the pot during the operation) and dumping the pot. The work is easy. The conditions under which it is done are comfortable, both as to temperature and atmosphere. Reports have shown a great reduction in liability to lead-poisoning in the works where the "H.-H." process has been introduced.

A new charge is started by kindling a small wood or coal fire on the grate, then throwing in a few shovelfuls of hot cal-

cines, and finally dropping in the regular charge of damp ore (plus the fluxes previously referred to). The charge is introduced in stages, successive layers being dropped in and spread out as the heat rises. At the beginning the blast is very low—about 2 oz. It is increased as the height of the ore in the pot rises, finally attaining about 16 oz. The operation goes on quietly, the smoke rising from the surface evenly and gently, precisely as in a well-running blast-furnace. While the charge is still black on top, the hand can be held with perfect comfort inside of the hood, immediately over the ore. This explains, of course, why the volatilization of silver and lead is insignificant. There is, moreover, little or no loss of ore as dust, because the ore is introduced damp, and the passage of the air through it is at low velocity. In the interior of the charge, however, there is high temperature (evidently much higher than has been stated in some descriptions), as will be shown further on. The conditions in this respect appear to be analogous to those of the blast-furnace, which, though smelting at a temperature of about 1,200° C. at the area of the tuyeres, suffers only a slight loss of silver and lead by volatilization.

At the end of the operation in the "H.-H." pot, the charge is dull red at the top, with blow-holes, around which the ore is bright red. Imperfectly-worked charges show masses of well-fused ore, surrounded by masses of only partly altered ore, a condition which may be ascribed to the irregular penetration of air through the charge, affording good evidence of the important part which air plays in the process. A properly-worked charge is tipped out of the pot as a solid cake, which, in falling to the ground, breaks into a few large pieces. As they break, it appears that the interior of the charge is bright red all through, and there is a little molten slag which runs out of cavities, presumably spots where the chemical action has been most intense. When cold, the thoroughly desulphurized material has the appearance of slag-roasted galena. Prills of metallic lead are visible in it, indicating reaction between lead sulphide and lead sulphate.

The columns of the structure supporting the pots should be of steel, since fragments of the red-hot ore dumped on the ground are likely to fall against them. To hasten the cooling of the ore, water is sometimes played on it from a hose. This

is bad, since some is likely to splash into the still inverted pot, leading to cracks. The cracked pots at certain works appear to be due chiefly to this cause, in the absence of which the pots ought to last a long time, inasmuch as the conditions to which they are subjected during the blowing-process are not at all severe. When the ore is sufficiently cold it is further broken up, first by driving in wedges, and finally by sledging down to pieces of orange size, or what is suitable for the blast-furnace. These are forked out, leaving the fine ore, which comes largely from the top of the charge, and is therefore only partially desulphurized. The fines are, therefore, re-treated with a subsequent charge. The quantity is not excessive; it may amount to 7 or 8 per cent. of the charge.

The breaking-up of the desulphurized ore is one of the problems of the process, the necessity being the reduction of several large pieces of fused, or semi-fused, material weighing two or three tons each. When done by hand only, as is usually (perhaps always) the practice, the operation is rather expensive. It would appear, however, to be a not difficult matter to devise some mechanical aids for this process—perhaps to make it entirely mechanical. When done by hand, a six-pot plant requires six men per shift sledging and forking. With 8-hr. shifts, this is 18 men for the breaking of about 60 tons of material, which is about $3\frac{1}{3}$ tons per man per 8 hr. With labor at 25c. per hour, the cost of breaking the fused material comes to 60c. per ton. It may be remarked, for comparison, that in breaking ore as it ordinarily comes, coarse and fine together, a good workman would normally be expected to break from 5 to 5.5 tons in a shift of 8 hr.

The ordinary charge for the standard converter is about 8 tons (16,000 lb.) of an ore weighing 166 lb. per cu. ft. With a heavier ore, like a high-grade galena, the charge would weigh proportionately more. The time of working off a charge is decidedly variable. Accounts of the operation of the process in Australia tell of charge-workings in from 3 to 5 hr., but this does not correspond with the results reported elsewhere, which specify times of from 12 to 18 hr. Assuming an average of 16 hr., which was the record of one plant, six converters would have capacity for about 72 tons of charge per 24 hr., or about 58 tons of ore, the ratio of ore to flux being 4:1. The loss in

weight of the charge corresponds substantially to the replacement of sulphur by oxygen, and the expulsion of carbon dioxide. The finished charge contains, on the average, from 3 to 5 per cent. of sulphur. This is about the same as the result achieved in good practice in roasting lead-bearing ores in hand-worked reverberatory furnaces; but curiously the "H.-H." product, in some cases at least, does not yield any matte, to speak of, in the blast-furnace, the product delivered to the latter being evidently in such condition that the remaining sulphur is almost completely burned off in the blast-furnace. This is an important saving effected by the process. In calculating the value of an ore, sulphur is commonly debited at the rate of 25c. per unit, which represents approximately the cost of handling and reworking the matte resulting from it. The practically complete elimination of matte-fall rendered possible by the "H.-H." process, however, may not be an unmixed blessing. There may be, for example, a small formation of lead sulphide, which causes trouble in the crucible and lead-well; and results in furnace difficulties and the presentation of a vexatious between-product.

It may now be attempted to summarize the cost of the converting process. Assuming the case of an ore assaying lead, 50; iron, 15; sulphur, 22; silica, 8, and alumina, etc., 5 per cent., let it be supposed that it is to be fluxed with pure limestone and pure quartz, with the aim to make a slag containing silica, 30; ferrous oxide, 40; and lime, 20 per cent. A ton of ore will make, in round numbers, 1,000 lb. of slag, and will require 344 lb. of limestone and 130 lb. of quartz, or, roughly, one ton of flux must be added to four tons of ore, wherefore the ore will constitute 80 per cent. of the charge. In reducing the charge to 3 per cent. of sulphur it will lose ultimately, through expulsion of sulphur and carbon dioxide (of the limestone), about 20 per cent. in weight, wherefore the quantity of material to be smelted in the blast-furnace will be practically equivalent to the raw sulphide-ore in the charge for the roasting-furnaces, but in the roasting-furnace the charge is likely to gain weight, because of the formation of sulphates. Taking the charge, which I have assumed above, and reckoning that as it came from the roasting-furnace it will contain 10 per cent. of sulphur, all in the form of sulphate, either of lead

or of lime, and that the iron be entirely converted to ferric oxide, in spite of the expulsion of the carbon dioxide of the limestone and the combustion of a portion of the sulphur of the ore as sulphur dioxide, the charge will gain in weight in the ratio of 1:1.18. This, however, is too high, inasmuch as a portion of the sulphur will remain as sulphide, while a portion of the iron may be as ferrous oxide. The actual gain in weight will consequently be probably not more than one-tenth. The following theoretical calculation will illustrate the changes:

	Raw Charge.		Semi-Roasted Charge.		Finished Charge.
ore	1,000 lb. Pb.	ore	1,154 lb. PbO.	ore	1,154 lb. PbO.
	300 lb. Fe.		428 lb. Fe_2O_3. (?)		428 lb. Fe_2O_3. (?)
	160 lb. SiO_2.		160 lb. SiO_2.		160 lb. SiO_2.
	100 lb. Al_2O_3, etc.		100 lb. Al_2O_3, etc.		100 lb. Al_2O_3, etc.
	440 lb. S.		300 lb. S.		68 lb. S.
flux	130 lb. SiO_2.	flux	130 lb. SiO_2.	flux	130 lb. SiO_2.
	344 lb. $CaCO_3$.		193 lb. CaO.		193 lb. CaO.
	....		450 lb. O.		....
	2,474 lb.		2,915 lb.		2,233 lb.
			10 per cent. S.		3 per cent. S.

Ratios:
2,474 : 2,915 :: 1 : 1.18.
2,915 : 2,233 :: 1 : 0.76⅔.
2,474 : 2,233 :: 1 : 0.90.

It may be assumed that for every ton of charge (containing about 80 per cent. of ore) there will be 1.1 tons of material to go to the converter, and that the product of the latter will be 0.9 of the weight of the original charge of raw material.

Each converter requires 400 cu. ft. of air per min. The blast-pressure is variable, as different pots are always at different stages of the process; but assuming the maximum of 16 oz. pressure, with a blast main of sufficient diameter (at least 15 in.) and the blower reasonably near the battery of pots, the total requirement is 21 h.p. The cost of converting will be approximately as follows:

Labor:	3 foremen at $3.20, .	$ 9.60
	9 men at 2.50, . .	22.50
Power:	21 h.p. at 30c., . .	6.30
Supplies, repairs and renewals .		5.00
Total, ...		$43.40 = 60c. per ton of charge.

The cost of converting is, of course, reduced directly as the time is reduced. The above estimate is based on unfavorable conditions as to time required for working a charge.

The total cost of treatment from the initial stage to the delivery of the desulphurized ore to the blast-furnaces, will be, per 2,000 lb. of charge, approximately as follows:

Crushing, 1.0 ton at 10c.,	$0.10
Mixing, 1.0 ton at 10c.,	0.10
Roasting, 1.0 ton at 63c.,	0.63
Delivering, 1.1 tons to converters at 12c.,	0.13
Converting, 1.1 tons at 60c.,	0.66
Breaking, 0.9 ton at 60c.,	0.54
Total,	$2.16

The cost per ton of ore will be $2.16 \div 0.80 = \$2.70$. Making allowance for the crushing of the ore, which is not ordinarily included in the cost of roasting, and possibly some over-estimates, it appears that the cost of desulphurization by this method, under the conditions assumed in this paper, is rather higher than in good practice with ordinary hand-worked furnaces, but it is evident that the cost can be reduced to approximately the same figure by the introduction of improvements, as, for example, in breaking the desulphurized ore, and by shortening the time of converting, which is possible in the case of favorable ores. The chief advantage, however, must be in the further stage of the smelting. As to this, there is the evidence that the Broken Hill Proprietary Co., after the introduction of the Huntington-Heberlein process, was able to smelt the same quantity of ore in seven furnaces that formerly required thirteen. A similar experience is reported at Friedrichshütte, Silesia.

This increase in the capacity of the blast-furnace is due to three things: (1) in delivering to the furnace a charge containing a reduced percentage of fine ore, the speed of the furnace is increased—*i.e.*, more tons of ore can be smelted per sq. ft. of hearth-area; (2) there is less roasted matte to go into the charge; (3) under some conditions the percentage of lead in the charge can be increased, reducing the quantity of gangue that must be fluxed.

It is difficult to generalize the economy that is effected in the blast-furnace process, since this must necessarily vary within

wide limits because of the difference in conditions. An increase of from 60 to 100 per cent. in blast-furnace capacity does not imply a corresponding reduction in the cost of smelting. The fuel-consumption per ton of ore remains the same. There is a saving in the power requirements, because the smelting can be done with a lower blast-pressure; also, a saving in the cost of re-working matte. Moreover, there will be a saving in other labor, in so far as portions thereof are not already performed at the minimum cost per ton. The net result under American conditions of silver-lead smelting can be determined closely only by extensive operations. That there will be an important saving, however, there is no doubt.

The cost of smelting a ton of charge at Denver and Pueblo, exclusive of roasting and general expense, is about $2.50, of which about $0.84 is for coke and $1.66 for labor, power and supplies. General expense amounts to about $0.16 additional. If it should prove possible to smelt in a given plant 50 per cent. more ore than at present without increase in the total expense, except for coke, the saving per ton of charge would be 70c. That is not to be expected, but the half of it would be a satisfactory improvement. With respect to sulphur in the charge, the cost is commonly reckoned at 25c. per unit. As compared with a charge containing 2 per cent. of sulphur there would be a saving rising toward 50c. per ton as the maximum. It is reasonable, therefore, to reckon a possible saving of 75c. per ton of charge in silver-lead smelting, no saving in the cost of roasting, and an increase of about 3 per cent. in the extraction of lead, and perhaps 1 per cent. in the extraction of silver, as the net results of the application of the Huntington-Heberlein process in American silver-lead smelting.

On a charge averaging 12 per cent. of lead and 33 oz. of silver per ton, an increase of 3 per cent. in the extraction of lead, and 1 per cent. in the extraction of silver would correspond to 25c. and 20c., respectively, reckoning lead at 3.5c. per lb., and silver at 60c. per oz. In this, however, it is assumed that all lead-bearing ores will be desulphurized by this process, which practically will hardly be the case. A good deal of pyrites, containing only a little lead, will doubtless continue to be roasted in Brückner cylinders, and other mechanical furnaces, which are better adapted to the purpose than are the lime-roasting

pots. Moreover, a certain proportion of high-grade lead-ore, which is now smelted raw, will be desulphurized outside of the furnace, at additional expense. It is comparatively simple to estimate the probable benefit of the Huntington-Heberlein process in the case of smelting-works which treat principally a single class of ore, but in such works as those in Colorado and Utah, which treat a wide variety of ores, we must anticipate a combination process, and await results of experience to determine just how it will work out. It should be remarked, moreover, that my estimates do not take into account the royalty on the process, which is an actual debit, whether it be paid on a tonnage-basis or be commuted in the form of a lump sum for the license to its use.

However, in view of the immense tonnage of ore smelted annually for the extraction of silver and lead, it is evident that the invention of lime-roasting by Huntington and Heberlein was an improvement of the first order in the metallurgy of lead.

In the case of non-argentiferous galena, containing 65 per cent. of lead (as in southeastern Missouri), comparison may be made with the slag-roasting and blast-furnace smelting of the ore. Here, no saving in cost of roasting may be reckoned, and no gain in the speed of the blast-furnaces is to be anticipated. The only savings will be in the increase in the extraction of lead from 92 to 98 per cent., and the elimination of matte-roasting, which may be reckoned as amounting to 50c. per ton of ore. The extent of the advantage over the older method is so clearly apparent that it need not be computed any further. In comparison with the Scotch-hearth bag-house method of smelting, however, the advantage, if any, is not so certain. That method already saves 98 per cent. of the lead, and, on the whole, is probably as cheap in operation as the Huntington-Heberlein could be under the same conditions. The Huntington-Heberlein method has replaced the old roast-reaction method at Tarnowitz, Silesia, but the American Scotch-hearth method, as practiced near St. Louis, is likely to survive.

A more serious competitor, however, will be the Savelsberg process, which appears to do all that the Huntington-Heberlein process does, without the preliminary roasting. Indeed, if the

latter be omitted (together with its estimated expense of 63c. per ton of charge, or 79c. per ton of ore), all that has been said in this paper as to the Huntington-Heberlein process may be construed as applying to the Savelsberg. The charge is prepared in the same way, the method of operating the converters is the same, and the results of the reactions in the converters are the same. The litigation which is pending between the two interests, Messrs. Huntington and Heberlein claiming that Savelsberg infringes their patent, will be, however, a deterrent to the extension of the Savelsberg process until that matter be settled.

The Carmichael-Bradford process may be dismissed with a few words. It is similar to the Savelsberg, except that gypsum is used instead of limestone. It is somewhat more expensive. because the gypsum has to be ground and calcined. The process works efficiently at Broken Hill, but it can hardly be of general application, because gypsum is likely to be too expensive, except in a few favored localities. The ability to utilize the converter-gases for the manufacture of sulphuric acid will cut no great figure, save in exceptional cases, as at Broken Hill; and, anyway, the gases of the other processes can be utilized for the same purpose, which is, in fact, being done in connection with the Huntington-Heberlein process in Silesia.

The cost of desulphurizing a ton of galena-concentrate by the Carmichael-Bradford process is estimated by the company controlling the patents as follows, labor being reckoned at $1.80 per 8 hr., gypsum at $2.40 per 2,240 lb., and coal at $8.40 per 2,240 lb.:

0.25 ton of gypsum,	$0.60
Dehydrating and granulating gypsum,	0.48
Drying mixture of ore and gypsum,	0.12
Converting,	0.24
Spalling sintered material,	0.12
0.01 ton coal,	0.08
Total,	$1.64

The value of the lime in the sintered product is credited at 12c., making the net cost $1.52 per 2,240 lb. of ore.

The low cost allowed for converting may be explained by the more rapid action that seems to be attained with the ores of Broken Hill than with some ores that are treated in North

America, but the low figure estimated for spalling the sintered material appears to be highly doubtful.

The theory of the lime-roasting processes is not yet well established. It is recognized that the explanation offered by Huntington and Heberlein in their original patent specification is erroneous. There is no good evidence in the process, or any other, of the formation of the higher oxide of lime, which they suggest.

At the present time there are two views. In one, formulated most explicitly by Professor Borchers, there is formed in this process a calcium plumbate, which is an active oxidizing agent. A formation of this substance was also described by Carmichael in his original patent; but he considered it to be the final product, not the active oxidizing agent.

In the other view, the lime, or limestone, serves merely as a diluent of the charge, enabling the air to obtain access to the particles of galena, without liquefaction of the latter. The oxidation of the lead sulphide is therefore effected chiefly by the air, and the process is analogous to what takes place in the Bessemer converter or in the Germot process of smelting, or perhaps more closely to what might happen in an ordinary roasting-furnace provided with a porous hearth, through which the air-supply would be introduced. Roasting-furnaces of that design have been proposed, and, in fact, such a construction is now being tested for blende-roasting in Kansas.

Up to the present time, the evidence is surely too incomplete to enable a definite conclusion to be reached. Some facts, however, may be stated.

There is already reaction to a certain extent between lead sulphide and lead sulphate, as in the reverberatory smelting-furnace, because prills of metallic lead are to be observed in the lime-roasted charge.

There is a formation of sulphuric acid in the lime-roasting, upon the oxidizing effect of which Savelsberg lays considerable stress, because its action is to be observed on the iron-work of the pipes in which it condenses.

Calcium sulphate, which is present in all of the processes, being specifically added in the Carmichael-Bradford, evidently plays an important chemical part, because not only is the sulphur trioxide expelled from the artificial gypsum, but also it is

to a considerable extent expelled from the natural gypsum, which is added in the Carmichael-Bradford process; in other words, more sulphur is given off by the charge than is contained by the metallic sulphides alone.

Further evidence that lime does, indeed, play a chemical part in the reaction is presented by the phenomena of lime-roasting in clay dishes in the assay-muffle, wherein the air is certainly not blown through the charge, which is simply exposed to superficial oxidation, as in ordinary roasting.

The desulphurized charge dropped from the pot is certainly much below the temperature of fusion, even in the interior, but we have no evidence of the precise temperature conditions during the process itself.

Pyrite and even zinc-blende in the ore are completely oxidized. This, at least, indicates intense atmospheric action.

The papers by Borchers,[7] Doeltz,[8] Guillemain[9] and Hutchings[10] may profitably be studied in connection with the reactions involved in lime-roasting. The conclusion will be, however, that their precise nature has not yet been determined. In view of the great interest that has been awakened by this new departure in the metallurgy of lead, it is to be expected that much experimental work will be devoted to it, which will throw light upon its principles, and, possibly, develop it from a mere process of desulphurization into one which will yield a final product in a single operation.

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[7] *Metallurgie*, vol. ii., pp. 1 to 6 (1905); *Engineering and Mining Journal*, vol. lxxx., pp. 398 to 400 (1905).

[8] *Metallurgie*, vol. ii., pp. 460 to 463 (1905); *Engineering and Mining Journal*, vol. lxxxi., pp. 175 to 176 (1906).

[9] *Metallurgie*, vol. ii., pp. 433 to 443 (1905); *Engineering and Mining Journal*, vol. lxxxi., pp. 470 to 471 (1906).

[10] *Engineering and Mining Journal*, vol. lxxx., pp. 726 to 728 (1905).

The Design of Blast-Furnace Gas-Engines in Belgium.*

BY PROFESSOR H. HUBERT, LIÈGE, BELGIUM.

(London Meeting, July, 1906.)

THE first attempts at direct utilization of blast-furnace gas in engines were made in 1895. For a considerable time the gas had been burnt in Cowper stoves for heating the blast for the furnace, and under the boilers which supplied steam to the blowing-engines, and others serving the furnaces. It was natural, therefore, that the idea of directly employing it in gas-engines should have occurred simultaneously to several engineers, notably to Lürmann and to Lencauchez, who had pointed out the blast-furnace as a powerful gas-producer. Nevertheless, nowhere had any attempt been made to apply it to this purpose up to the end of 1894, when Thwaite proposed it to Mr. James Riley, of the Glasgow Iron & Steel Company.

About the same time investigations were being made in Belgium and in Germany, independently of Thwaite's experiments, which were not generally known on the Continent.

The industrial world, which up to that time had hardly favored the idea, had thus been gradually prepared to receive it. The gas-engine, long restricted to small sizes and dependent upon the use of an expensive fuel obtainable only in large centers, now began to make headway.

At the Paris Exhibition of 1889 two engines of 100 h.p. were shown and excited much interest among engineers. One had four cylinders, and was made at the celebrated works of the Deutz Company, and the other was a single-cylinder engine, exhibited by two French designers, Messrs. Delamare-Deboutteville and Malandin.

In the meanwhile the design of gas-producers had made important progress, completely freeing the new engine from its

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dependence on the gas-works, enabling it to be installed anywhere, and to realize to the full extent its economic value by supplying it with a cheaper fuel.

In 1892 Delamare installed at the Moulins Leblanc Works at Pantin a four-cycle, single-acting, single-cylinder engine, using producer-gas, and developing 220 b.h.p., with a consumption of about 1 lb. of coal per b.h.p. per hr. Despite the difficulties met with in this bold attempt, it showed the possibility of economically producing high power with poor gas.

The time had now arrived for engineers to pay attention to the use of gas from blast-furnaces, which, although not of great heating value, was less costly; and was the more suitable on account of the progress which had been made in the design and working of blast-furnaces, the proportionally lower consumption of coke, and, as a result, the marked reduction in the relative quantity of combustible gases, which only sufficed, with difficulty, to heat the blast and to produce the steam required about the furnaces. Finally, the progress of the science of heat had brought to light the causes of the low thermal efficiency of the steam-engine, and notably of the loss resulting from the employment of boilers.

Therefore it is not surprising that the idea of dispensing with the boiler and burning the blast-furnace gases directly in the engine occurred, nearly simultaneously, in three countries, where metallurgical industry had made great progress.

To Messrs. Bailly and Kraft, of the Cockerill Company, belongs the honor of being the first in the field in Belgium. The patent taken out by the Cockerill Company for this new application was dated May 15, 1895, and the first trials were made at the end of that year. They were made with a Simplex engine of 8 h.p., in which it had only been sought to reduce the clearance-space, in order to increase the compression and to facilitate the ignition of the mixture. The gas-cleaning was very imperfect, and was carried out simply by passing it through two scrubbers, 4 m. in height.

This engine displayed perfect elasticity, and adapted itself to the variations of composition, pressure and temperature of the gases, giving an efficiency of 77 per cent.

I have described[1] the results of the first trials, and the conditions necessary for the direct use of blast-furnace gases, showing that a plant producing 100 tons of pig-iron per day was able to furnish about 18,000 cu. m. of gas per hr. with a calorific value of 1,000 calories. Taking into consideration that half this volume is available, and allowing for an efficiency of only 20 per cent. in the engines, I showed that it would be possible to obtain from these gases about 3,000 horse-power.

The small trial engine consumed about 5 cu. m. per b.h.p., which reduces the preceding figures to 1,800 h.p.; but I foresaw at the time that this consumption ere long would be greatly reduced, and that blowing-engines driven by gas would be built. I also foresaw that, by disposing of the great surplus motive power, the blast-furnace would ultimately become a center for the production of energy for works surrounding, the boilers of which it would gradually supersede.

This remarkable progress was described by Mr. E. P. Martin, President of the Iron and Steel Institute, in his Presidential Address of 1897.[2]

At the meeting of the Institute on May 3, 1898, M. A. Greiner discussed[3] the results published up to that time, which included my paper of February, 1897, a note by Galbraith and Rowden on November 18, 1897, one by Lencauchez on November 8, 1897, and another by Lürmann on February 27, 1898.

Mr. Greiner combated the objections which had been specially raised against this new method of employing gas by German metallurgists at the Düsseldorf meeting, and gave reasons for his belief that the consumption would be reduced below 4 cu. m. per b.h.p. per hr., and that the blast-furnace would be able, by superseding the steam-boiler as an intermediary in the production of motive power, to place at the disposal of the engineer 20 h.p. per ton of pig produced daily.

Experience soon verified these forecasts. The Cockerill Company have constructed, with the collaboration of Mr. Delamare, a four-cycle, single-cylinder engine of the Simplex type, which, in the 24-hr. trials, at which I had the honor to

[1] *Annales des Mines de Belgique*, 1897.

[2] *Journal of the Iron and Steel Institute*, vol. li., No. 1, pp. 19 to 40 (1897).

[3] *Journal of the Iron and Steel Institute*, vol. liii., No. 1, pp. 21 to 31 (1898).

collaborate with Professor A. Witz, gave 1 b.h.p. for an average consumption of 3.329 cu. m. of a gas possessing a calorific power of 981 calories—say 3,266 calories. (This figure has since been reduced to 3,162.)

The principal dimensions of this engine are: diameter of cylinder, 0.8 m.; stroke of piston, 1 m.; rev. per min., 105; i.h.p., 213; b.h.p., 182.

The construction of this engine is worthy of note. The cylinder proper is cast with a breech carrying at its lower portion the exhaust-valve, and at the back a cylindrical prolongation in which the admission-valve is placed. This breech or cylinder-head has its own water-jacket, and is provided towards the front with flanges bolted to the cylinder-jacket, which is a part of the cylinder bayonet casing.

The shaft is not cranked, and carries a heavy fly-wheel. The piston is made in one piece, and is not chilled. The compression does not exceed 7 kg. The sparking is effected by Delamare's system, in which a succession of sparks, produced by a Ruhmkorff coil, is emitted from a slide-valve on the back of the cylinder when its opening comes opposite to an orifice bored in the back. The movement of the sliding-valve, and similarly of the other valves, is made by a crank and by cams keyed on to an auxiliary shaft, parallel to the cylinder and revolving at half the speed of the main shaft. The governing is effected by the hit-and-miss arrangement by means of Delamare's air governor.

Starting is effected by turning the fly-wheel, by a hand-wheel, and by admitting a charge of carbureted air, the explosion of which starts the engine.

The success of this engine, which worked perfectly without the gas being cleaned as effectively as is now done, and is still running after being eight years in service at Cockerill's, encouraged them to build a much more powerful type of engine capable of directly operating a blowing-apparatus of 600 h.p., and consequently of liberating the blast-furnace from its dependence upon the boiler.

Though the enterprise was considered rash, they still went on with the attempt, which was logically justified, seeing that in modern steam-driven blast-furnace installations the gas produced is only just sufficient for the requirements of the fur-

naces. To procure gas in excess it was necessary to commence by replacing the existing engines by the more economical gas-engines, for by such means only would gas be available. It is necessary to commence with gas-driven blowing-apparatus. A motor of this description attracted much attention at the Paris Universal Exhibition of 1900. Another, coupled with its blowing-apparatus, in the blast-furnace department of the Cockerill Company, and started up on November 20, 1899, was submitted by me to a series of trials on March 20 and 21, 1900.

The features of this remarkable engine were as follows: regulation by the method of "hit and miss," that is to say, suppression of an admission of gas complete; diameter of cylinder, 1.300 m.; stroke of piston, 1.400 m.; rev. per min., 94.4.

By brake tests.	Indicated horse-power, ...	786	with 89 p. ct. admission and 11 p. ct. "hit and miss" by the governor.
	Effective brake horse-power, . . .	575	
	Consumption per indicated horse-power hour,	2.556 cu. m. or 2,515 calories.	
	Consumption per brake horse-power hour, .	3.495 cu. m. or 3,440 calories.	
Tests with the blowing-apparatus.	Number of revolutions per minute, . .	93	
	Indicated horse-power, ...	886.5	full charge without "hit and miss."
	Effective brake horse-power, . . .	725	
	Consumption per indicated horse-power hour,	2.334 cu. m or 2,343 calories.	
	Consumption per brake horse-power hour, .	2.853 cu. m or 2,864 calories.	

The method of construction of the 200-h.p. motor had generally been retained, saving that the main bearings were separated from the cylinder-casing, and were connected by four strong screwed steel stay-bolts, giving easy access to the piston. The shaft was cranked and rested on three bearings to support the fly-wheel, which weighed 33 tons. The piston-rod traversed the back space in a stuffing-box.

The admission-valves were retained below like those of the exhaust. The admission of gas was carried out by separate valves placed in a valve-box, separated from the cylinder by a third valve called the mixture-valve. The methods of working the valves and those of the ignition slide-valve were retained. The regulation was carried out by "hit and miss." The pressure attained 9 kg. per sq. cm. The circulation of water extended to the head and to the piston-rod itself, to which the water penetrated by means of flexible pipes, which

adapted themselves to its movement. The exhaust-valve was also cooled. This was done with the object of preventing the ignition of the mixture by the dust, which, combining with the products of the decomposed oils on the piston or in the recesses of the explosion-chamber, might form concretions retaining a temperature high enough to ignite the gases.

The arrangement of all the valves at the under-side of the cylinder was such as to facilitate the sweeping-out of the dust and decomposed oil, and to allow these large engines to work equally as well as the 200-h.p. engines without having recourse to a more perfect gas-cleaning process. This hope was ill-founded. It became necessary to interpose, between the blast-furnace and the large engines of this class, apparatus capable of reducing the dust held in suspension by the gas to 0.02 g. per cu. m. The means now used in Belgium are centrifugal fans with water injection, and Theisen, Bian, and Zschocke apparatus. The latter are not, strictly speaking, purifiers; they are rather coolers. As they are not the invention of Belgian engineers, and as they will be made the subject of another paper, it is not necessary to deal further with them.

As is well known, the novel idea of the Cockerill Company was vigorously discussed by engineers, who saw therein an economic mistake, and maintained that it was better to divide the power between two or four cylinders. The designers, nevertheless, knew perfectly well that they could obtain in this way, for a 600-h.p. engine, a more regular and perhaps more economical engine. They had already, however, studied the two-cylinder tandem types of 600 and 1,200 h.p. One of Cockerill's licensees, Messrs. Breitfeld, Danek of Prague, had since 1901 constructed a four-cylinder, double tandem engine of 600 h.p. giving remarkably even running; but Messrs. Cockerill wished to demonstrate that it was practically possible to develop 600 h.p. by means of a single cylinder alone, single acting and of four cycles, and consequently to construct engines developing up to 2,500 h.p. without exceeding four cylinders. In addition to this they were, moreover, anxious also to improve the governing, by applying to these large engines the principle of variable admission in lieu of the hit-and-miss governing, which required the use of heavy fly-wheels, and was

not well suited for producing alternating electric currents, and needlessly strained the engine when it had to run continuously with reduced loads.

From 1901 they realized with M. Delamare that it was essential to obtain a variable-admission motor, an air-governor, or else a centrifugal force, causing the double air- and gas-valves to open from the commencement of the suction-stroke, but determining the closure earlier in the stroke as the power to be developed becomes lower.

In this manner the mixture admitted possessed the composition most favorable for complete combustion, but the volume admitted to the cylinder varied, and with it the compression.

The ingenious mechanism which realized this mode of operation I have described elsewhere;[4] it was applied to a single-cylinder motor of 200 h.p. of the same dimensions as that of 1898.

The trials to which it was submitted in November and December, 1901, established beyond doubt a consumption varying between 3.318 and 3.455 cu. m. per b.h.p. per hr. for full load, the calorific value being from 914 to 1,017 calories. The expenditure in calories per h.p. varied between 3,172, and 3,434, and has been on an average nearly 3,298; practically the same as that of 1898. At half-load it was, on an average, 4,320 calories, and at quarter-load 7,406 calories.

About the same time (1902) the Cockerill Company produced another engine, designed to give greater regularity with smaller dimensions—viz., the double-acting engine.

It was well known that the first industrial gas-engine, that of Lenoir, was double-acting, but the success of the Otto four-cycle and single-acting engine had, for a long time, relegated to the background all other types of engines. Nevertheless, M. Letombe, at the Brussels Exhibition, showed a four-cycle and double-acting engine.

The Körting Company exhibited at Düsseldorf a powerful two-cycle, double-acting engine, which attracted much attention. The long-standing prejudice against the adoption of this system was thus broken down.

The direct driving of the blowing-apparatus from the piston-

[4] *Revue Universelle des Mines*, vol. lix., pp. 273 to 329 (1902).

rod of the engine had accustomed Cockerill's engineers to the adoption of a stuffing-box at the back end of the cylinder. They were, therefore, naturally disposed to adopt double action, which enabled them to considerably reduce the size of the cylinder, and consequently approach large powers more easily, thus insuring more steady running with a lighter fly-wheel.

They retained, firstly, the general arrangement of the single-acting motor, which up to that time they had constructed, notably the disposition of the inlet- and exhaust-valves underneath the cylinder. However, from that time they introduced an important modification. The cylinder-liner, with its jacket, constituted a part independent of the two cylinder-heads. Each of these carried a stuffing-box, through which the water-cooled piston-rod worked, and an extension downwards of the combustion-chamber, in which were installed the valve, which simultaneously admits air and gas and actuates the exhaust-valve. The actuating mechanism of the valves was also modified, without departing from the system of variation of the admission, consisting of cutting off the air and gas supply simultaneously. This system had the advantage of preserving the composition of mixture most favorable to complete combustion, but it had the inconvenience of diminishing, to some extent, the compression as the charge decreased. This diminution reduced the economic efficiency of the engine in the case of light loads, and also when it happened that the gas was very poor it spoilt the ignition and caused misfires, which altogether upset regularity and economy.

This trouble becomes very marked in motors driving dynamos, which very often work with reduced loads, and where economy is a greater consideration than in blowing-engines. Therefore, no time was lost in introducing another system of variation, consisting of air-admission to the cylinder during the whole piston-stroke, and only allowing the gas to enter during the last portion of the stroke by the governor varying the moment at which this admission commences. In this way invariable compression is secured.

It is true that when the mixture is modified it becomes poorer and poorer; but it should be noted that gas is introduced at the back-end of a cylinder already partly filled by a

volume of air which follows the piston. Although it is impossible absolutely to rely upon retaining the exact stratification characteristic of the Otto cycle, there persists, nevertheless, an undoubted stratification of mixture, the richest strata remaining at the back-end of the cylinder, close to the igniter.

The sparks then impinge on the explosion-mixture, which, being strongly compressed, insures that the ignition is readily transmitted to the whole volume. It will be seen that the gas-admission valve should be able to move independently of that giving air admission.

In engines of this system the two valves are superposed, the air arriving by a casing which surrounds the gas-passage, and the valve-spindle passing through the gas-valve, which is hollow. The placing of the valves in an antechamber of the combustion-chamber, leading to a tubular combustion-chamber, evidently assists the stratification of which mention has been made, and consequently the ignition of weak charges, but it resulted in cylinder-heads of unsymmetrical form, which created difficulties at Cockerill's works, as it had already done elsewhere.

The unequal contraction of the metal of the various parts of the cylinder-head caused great stress, which, added to the already high stresses, due to the explosion and to the heating, has occasionally brought about the fracture of the cylinder-heads, even when they have been replaced by steel castings.

This circumstance decided the makers of large engines to revert to the symmetric arrangement of the valves, which is customary in steam-engines, where the inlet-valve is placed on the top of the cylinder and that of the exhaust underneath, and thus to obtain an arrangement which lends itself well to expansion, and which, moreover, facilitates access to the valves.

This arrangement has been obtained in different ways by manufacturers, notably at the works of Deutz and Nürnberg and at Seraing. At the Cockerill Company's works the covers are no longer attached to the central body by studs screwed into it, but joined by tie-bolts bolted to flanges on these covers.

These bolts are thus subjected to tension, and, similarly, the body of the cylinder is subjected to a compression stress of the kind which best suits such metal. This arrangement, shown in Fig. 1, is patented. The frame is formed of two box-

girders carrying the cylinder. These girders are joined by tie-bolts to others that contain the slides and carry the crank-shaft bearing. The piston is composed of two halves with double walls, each half permitting water-circulation, the two halves being bolted together with an india-rubber joint.

The water cooling is effected at a pressure of from 3.5 to 5 kg. per sq. cm. to avoid water-hammering in the piston and its rod. The water, furnished to the latter by a duct fixed at one end, passes through the rod and the two halves of the piston, and goes out at the back by another duct.

The ignition is effected by means of one or two high-tension magnetos, through fixed sparking-plugs. These magnetos do away with the necessity for a source of electricity external to the motor. The compression has been successively increased up to 14 kg. per sq. cm. The starting is effected by means of air, compressed to 10 atmospheres by a special compressor, and retained in a reservoir in sufficient quantity to enable the engine to revolve several times.

The assembling of the parts, which constitutes one of the features of this motor, must now be described. As shown in Fig. 2, the usual arrangement of the auxiliary side shaft, parallel to the cylinder-axes, and actuated by the main shaft by means of gearing in the ratio of 1:2, is retained. On this auxiliary shaft, B, are keyed the cams, C, which actuate the opening of the inlet- and exhaust-valves.

The exhaust-valve, E, opens a little before the end of the stroke, but always at the same moment in each stroke; it is worked by the cam, C, by means of a lever and a rod carrying a roller at its lower end. The exhaust-valve is hollow, and likewise the spindle, to permit of the circulation of cooling-water.

The inlet distributor consists of two valves: one for gas, the other for air and the mixture. The spindle of the latter passes through the hollow spindle of the gas-valve. Both are constantly brought back to their seats by springs placed in a box above. The air-valve is double. It consists of a solid single-seated flap-valve opening downwards, and a hollow perforated sleeve-valve, L. These two parts are attached to the same spindle and move simultaneously.

From a little in advance of the commencement of the suction

stroke, the double valve is lowered by the lever, D, actuated by the rod, *t*, at the lower end of which works the cam, C. By this movement the openings of the sleeve-valve, L, are brought opposite the ports formed in the casing, by which the incoming air arrives. The latter is thus drawn in from the commencement of the piston-stroke and throughout its entire duration.

To the lever, D, is attached a spindle, F, which works a second lever, K, the function of which is to lower the double-seated gas-valve, M. If this spindle was unalterable in length, the two levers, D and K, would work simultaneously, and the gas would pass into the cylinder simultaneously with the air. But the spindle, F, is composed of two parts, of which one carries the piston and the other the cylinder of a dash-pot, P.

The upper part is thus able to rise up, expanding the air in the cylinder, provided that the lower portion remains fixed, and in this case the gas-valve does not open. The end of the lever is locked during the period of exhaust by a finger, H, which keeps it from moving as long as the finger remains vertical. If this part is moved to the right the lever, K, now becomes free, but the descent of the spindle of the mixing-valve, A, compresses the spring, R, placed upon that of the gas-valve, M, as long as this is immovable. Immediately K is unlocked, this spring is able rapidly to lower M, causing the valves to resume their original distance. The shock is deadened by the dash-pot, and the spindle, F, likewise resumes its normal length.

From this time forward the gas penetrates, with the air, into the cylinder, constituting the explosive mixture, which will be afterwards compressed, and ignited by the spark. In order, therefore, to enable the admission of gas to be varied, it suffices to operate at the moment the lever, K, is unlocked. This instant is determined by the governor in the following manner: The finger, H, is articulated on a shaft, N, from whence it is prolonged to engage a spring, which tends to keep it constantly locked with K.

Its lower extremity, I, carries a roller, on which is taken the thrust of the cam, V, fixed on the oscillating shaft, X. It is this cam which produces the unlocking of K, by pushing against I, and turning aside H, despite the resistance of *a*.

The oscillating movement of X is produced by a rod, *b*, which operates the crank keyed on X. The rod is attached to one

end of a transmitting-lever, Z, the other end of which is worked by an eccentric, O, keyed upon the auxiliary shaft, B. The lever, Z, however, constitutes an extension of the roller of an eccentric-strap, T, of which the sheave is keyed on the spindle, W. This carries a crank joined by a rocking-beam to the centrifugal governor-socket. The movements of the governor-socket thus modify the position of the shaft, W, and consequently the center around which Z oscillates. This results in an alteration in the proportion of the two arms of this lever, and, as a result, the length of travel of the connecting-rod, *b*, and increases the oscillation of X. It may be seen, therefore, that the moment at which the gas-valve opens is thus determined by the governor, which retards or advances it as the speed of the engine increases or diminishes.

When, as a result of the rotation of the shaft, B, the cam, C, ceases to operate the connecting-rod, *t*, the springs mounted upon the spindles of the two valves bring the latter back to their seats. It will be seen from this that, as a result of the clearance between the parts, or from the expansions, the gas-valve closes first, leaving open the mixture-valve. This hinders the compression. To avoid this inconvenience the spring of the gas-valve is made weaker than the other, so that it is able, despite the arrest of the former, to replace the latter on its seat, by slightly compressing the first spring.

To this somewhat lengthy description of what is really a simple mechanism, which works perfectly, it may be added that the exhaust commences about 15 per cent. in advance, and is prolonged a little beyond the finish of the stroke, during which time the admission has slightly commenced. The result is that the exhaust still continues. At this juncture the burnt gases have attained a high velocity, which causes a powerful suction in the cylinder.

The atmospheric air rushes violently into the explosion-chamber, completely sweeping out the burnt gaseous residuals, which otherwise would hang in the vicinity of the sparking-plug and tend to spoil the ignition of the mixture.

The application of this ingenious mechanism confers on the double-acting engine very even running, allowing alternators of 50 periods easily to be coupled in parallel.

This type, studied by the Cockerill Company since 1904, was

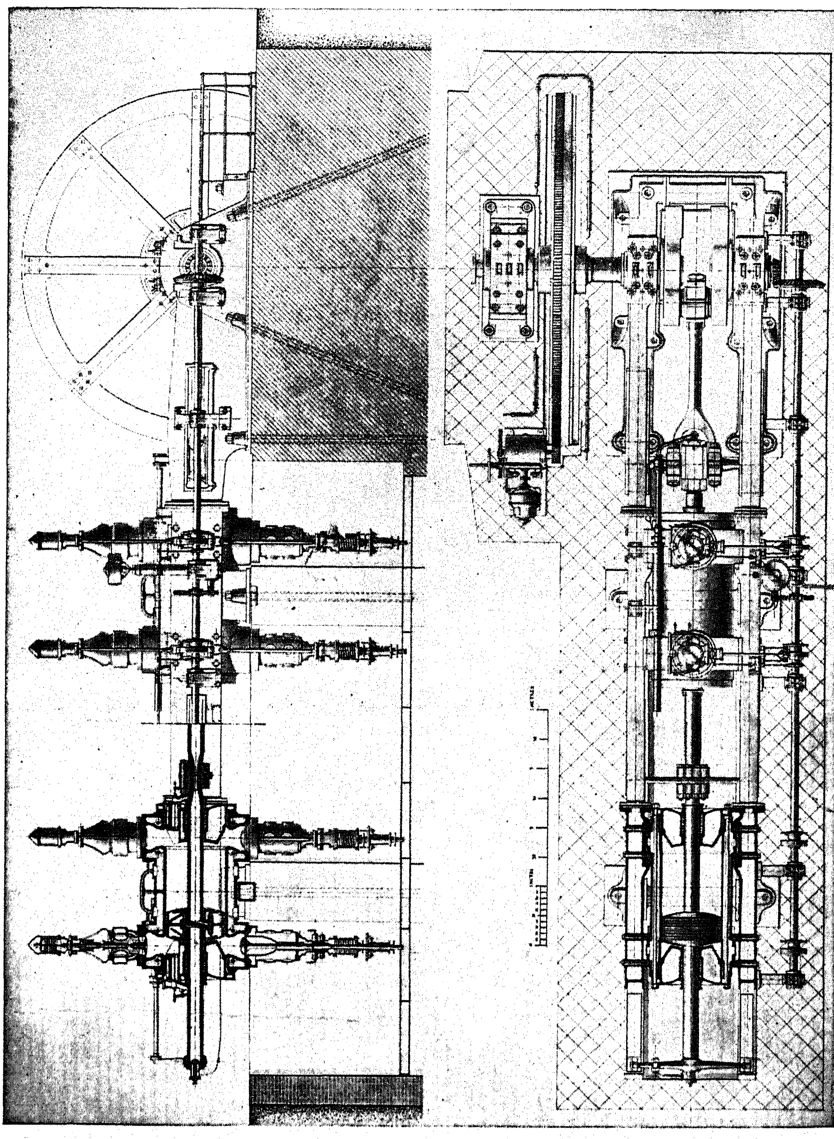

FIG. 1.—VALVE ARRANGEMENT OF THE COCKERILL ENGINE.

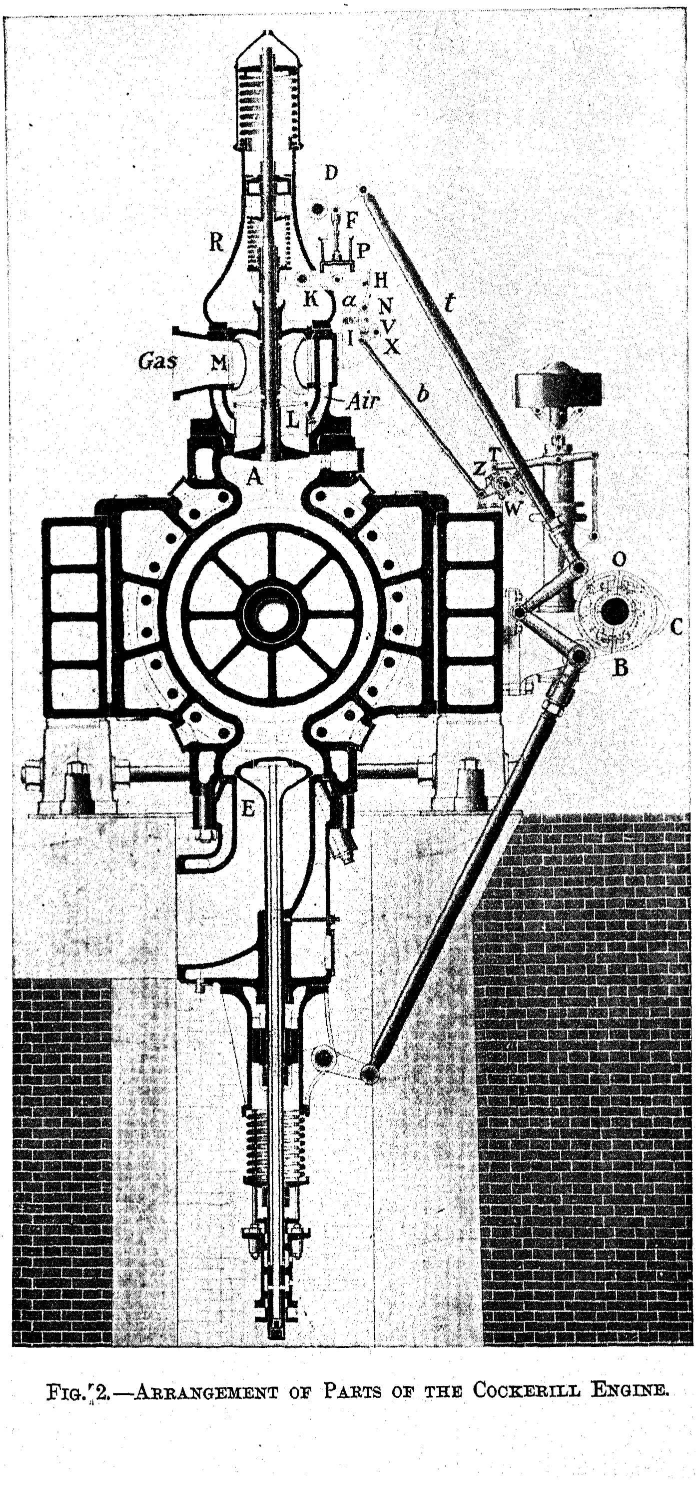

FIG. 2.—ARRANGEMENT OF PARTS OF THE COCKERILL ENGINE.

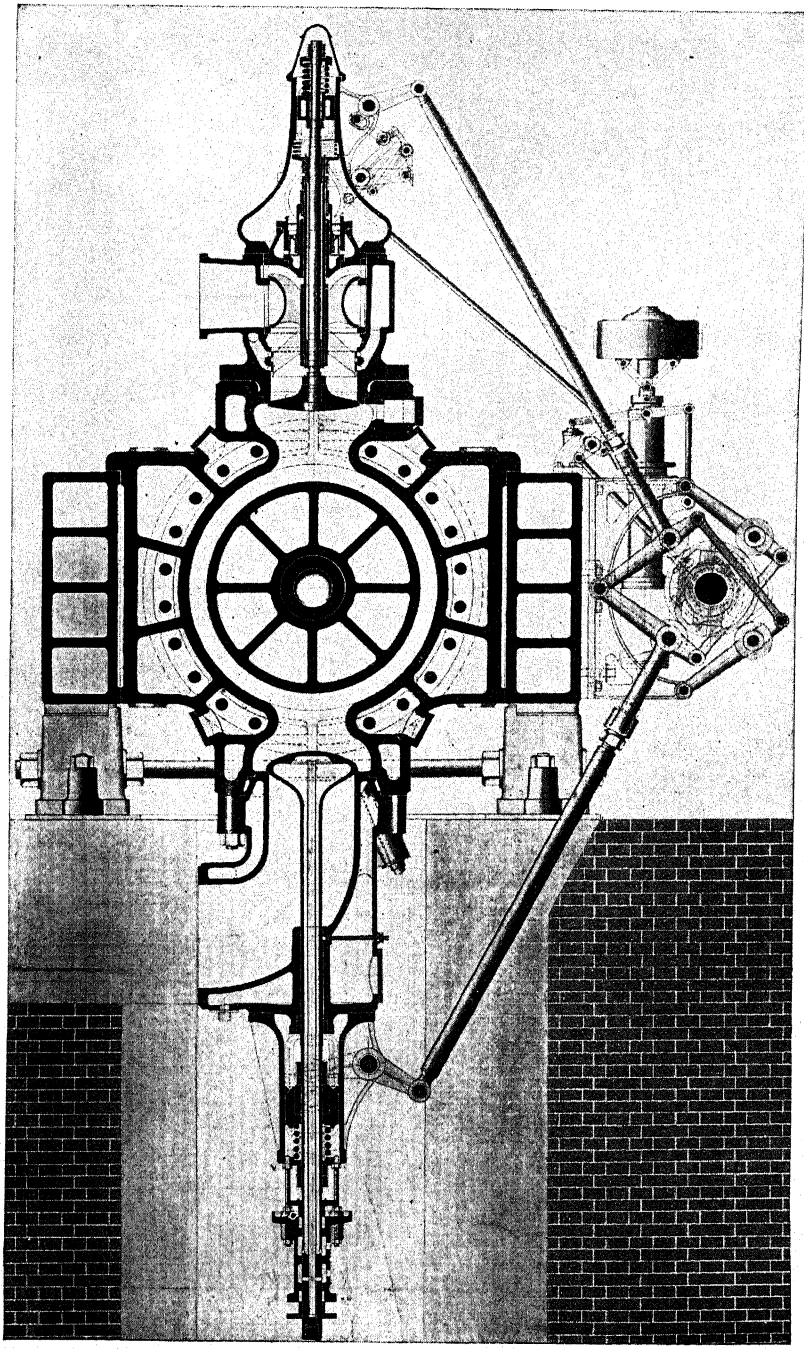

FIG. 3.—SPECIAL VALVE ARRANGEMENT OF THE COCKERILL ENGINE.

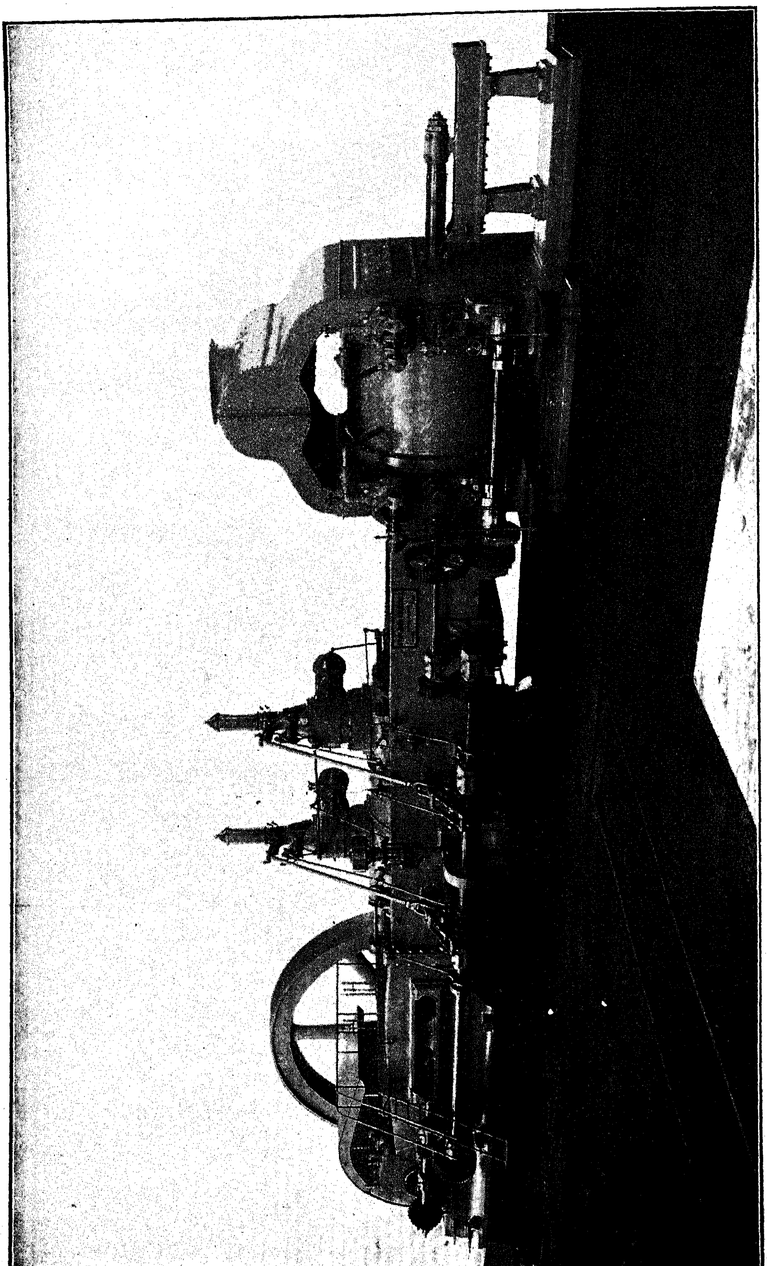

Fig. 4.—New Type of the Cockerill Engine.

put in service on a two-cylinder tandem-engine of 1,400 h.p., installed in the electricity department of the company.

The pistons are 1 m. in diameter by 1.1 m. stroke. The shaft makes about 100 rev. per min., and carries a fly-wheel of 26 tons, and drives a continuous-current dynamo.

Engines of this class are made of 500 h.p. (600 mm. × 800 mm. × 135 rev. per min.), of 750 h.p. (750 mm. × 900 mm. × 100 rev. per min.), and of 1,400 h.p. (1,000 mm. × 1,100 mm. × 100 rev. per min.).

The Cockerill Company have also constructed a single-cylinder engine of 575 h.p. (1 m. × 1.100 m. × 80 rev. per min.), one of 1,000 h.p. (1.150 m. × 1.250 m. × 80 rev. per min.), and one of 1,300 h.p. (1.300 m. × 1.400 m. × 80 rev. per min.

However, for this special purpose, the constant compression, necessary for the economical production of electricity, might become troublesome when the engine is of the single-cylinder type.

It happens, indeed, that the blowing-apparatus may be perceptibly slowed down, and then it might be that the centrifugal force of the fly-wheel would be insufficient for passing the dead-center at the time of compression, especially if a miss-fire had just previously taken place.

It is advantageous, in this case, to do away with constant compression and revert to variable quantity admission, of a mixture of constant composition (Fig. 3 shows this arrangement). This is a little more complicated than the other, and has been worked out for two types—one of 575 h.p. (1 m. × 1.100 m. × 80 rev. per min.), and the other of 1,300 h.p. (1.300 m. × 1.900 m. × 80 rev. per min.).

It is interesting to be able to compare the economy of the new type of engine (Fig. 4) with those of the preceding types already referred to.

The Cockerill Company instructed me to undertake, with Professor Witz, detailed tests on the 1,400-h.p. two-cylinder double-acting and tandem engine installed in their electric service. These tests were made on January 9 and 10, 1906. Arrangements were made for measuring the consumption, a bell, holding about 300 cu. m., being provided for this purpose.

As the engine working at full load consumes about 2.5 cu. m.

TABLE I.—*Results of Tests on the 1,400-h.p. Two-Cylinder Double-Acting and Tandem Engine.*

Description of Test.	Indicated Horse-power.	Brake Horse-power.	Electrical Horse-power.	Mechanical Efficiency, Per Cent.	Electrical Efficiency, Per Cent.	Total Efficiency, Per Cent.	Consumption per Hour and per Horse-power (*cheval*) in Cubic Meters, at 0.02760 Millimeter per			Highest Calorific Power by the Witz Bomb per 3 Meters. Calories.*	Calories Used per Hour and per			Thermal Efficiency Per Cent.	Heat Carried Away by	
							Indicated Horse-power.	Brake Horse-power.	Electrical Horse-power.		Indicated Horse-power.	Brake Horse-power.	Electrical Horse-power.		The Water of Circulation, Per Cent.	The Exhaust Gas.
½ load......	1250.48	986.00	917.5	78.7	93.1	73.3	2.513	3.187	3.425	979	2160.2	3120.1	3353.1	25.82	16	Not measured.
¾ load......	1463.88	1332.7	1250.7	91.10	93.8	85.4	2.313	2.540	2.707	963	2227.4	2446.0	2606.8	28.52	16.2	
Full load.	1569.45	1466.15	1377.45	93.41	93.9	87.76	2.247	2.406	2.560	983	2208.8	2365.1	2516.5	28.77	20.2	
Full load.	1607.74	1494.8	1404.7	92.97	94.0	87.4	2.325	2.497	2.657	943	2192.5	2354.7	2505.6	28.98	16.2	
Overload..	1755.06	1581.9	1487.7	90.00	94.0	84.7	2.155	2.392	2.542	988	2129.1	2363.3	2511.5	29.84	20.2	

* This is greater than that given by the Junkers calorimeter.

per h.p. per hr., the bell was only able to feed it for a little more than 5 min. It was then raised again by the gas being pumped into it by an electric rotary-pump. In the interval, the opening of gas-valves allowed the engine to take gas directly from the main, from which these gas-valves isolated it during the time of the trials.

The indicated work (i.h.p.) was determined by a large number of diagrams, taken by two observers. The useful work (effective horse-power, b.h.p.) was ascertained by means of the electric energy developed by a dynamo, of which the output had been carefully tested in advance for a series of powers. Samples of the gas were taken, for the determination of the calorific power, by the Witz bomb, in the laboratory of Professor Witz at Lille University. The quantities of water used in cooling the different parts of the engine were measured, as well as their temperatures. It was not found practicable to take the gas at the outlet, because the water injected into the exhaust vaporized instantly and lowered the temperature.

The detailed results of these tests will be published later; but the principal elements, enabling the progress made since the commencement of this novel application of blast-furnace gases, and the enormous economy it has brought about, to be measured, are given in Table I.

It is interesting to compare the results obtained in these trials with those obtained previously with engines made by the same company.

The figures, given in Table II., are significant. In comparing the trials of 1906 with those of 1900, which gave results which were considered excellent at that time, there was found a diminution of 15 per cent. of calories used per i.h.p. For the b.h.p. the reduction attains 31.4 per cent. Finally, the thermal efficiency has been increased 18.4 per cent. The advance made is thus very considerable. The advantage obtained, as compared with the employment of steam, is by no means the least interesting of facts brought out by this investigation. Admitting, however, that a 70-per cent. efficiency is obtained by burning the gases under boilers, which is considered satisfactory, and that the steam raised amounts, according to Carnot's cycle, to an efficiency of 40 per cent. (between 200° and 10°, which the cycle of Rankine makes 77 per cent. of that of Car-

TABLE II.—*Results of Former Tests Compared with Tests on 1,400-h.p. Two-Cylinder Double-Acting and Tandem Engine.*

No.	Description.	Date of Trials.	Power.		Consumption of Gas.		Consumption of Heat.		Thermal Efficiency.
			I. H. P.	B. H. P.	I. H. P.	B. H. P.	I. H. P.	B. H. P.	Per Cent.
1.	*8 Horse-Power Engine.*	1896	5.26	4	4 03	5.30	4030	5300	15.77
2.	*200 Horse-Power Engine.* Single cylinder, single-acting, constant admission.	July 19–20, 1898	213.9	181.82	2 830	3.329	2775.8	3265.7	22.9
3.	*600 Horse-Power Engine.* Single cylinder, single-acting, constant admission.	March 20, 1900	825.8	670.0	2.560	3.156	2520.1	3106.8	25 2
4.	*200 Horse-Power Engine.* Single cylinder, single-acting, variable admission.	Dec. 2, 1901	246.9	215 3	2.981	3.418	2766.3	3172	23.0
5.	*1400 Horse-Power Engine.* Double-acting, tandem, variable admission.	Jan. 9 & 10, 1906	1755.06	1581.9	2.155	2.392	2129.1	2363.3	29.84

not), and finally that one could succeed in obtaining 80 per cent. of the Rankine cycle, the thermal efficiency becomes

$$0.70 \times 0.40 \times 0.77 \times 0.8 \times 100 = 17.25$$

which is lower, by 42 per cent., than that of the motor above described.

It has been assumed, earlier in the paper, that a blast-furnace producing 100 tons of pig-iron per day yields about 9,000 cu. m. of gas at 1,000 calories, available for the production of power.

Under these conditions, by obtaining 2,450 h.p. with the steam-engine, and 4,220 h.p. by using the gas direct in gas-engines, there remains a difference of 1,770 h.p. in favor of this new application over the results obtained by a first-class steam-plant of the usual (boiler) type. Compared with the usual boiler and steam-engine plant, the gain by the use of a gas-engine may reach 2,500 h.p., or 25 h.p. per ton of daily production.

These figures explain the success obtained in metallurgy by the direct use of blast-furnace gas.

Besides the Cockerill Company there is in Belgium but one other company (the Société de Saint Leonard) which has un-

TABLE III.—"*Cockerill*" *Type Gas-Engines at Work or Building.*

I. H. P.	Type.	Driving.	Built for	Working Since
260	Single Cyl.	Dynamo.	Société Anonyme John Cockerill, Seraing, Belgium.	Mar., 1898
260	Single Cyl.	Dynamo.	Roechlingsche Eisen- u. Stahlwerke, Carlshutte.	Jan., 1899
800	Single Cyl.	Blowing Cyl.	Société Anonyme John Cockerill, Seraing.	Nov., 1899
800	Single Cyl.	Blowing Cyl.	Differdinger Hutte (Differdange).	Nov., 1899
800	Single Cyl.	Blowing Cyl.	Differdinger Hutte (Differdange).	Apr., 1900
800	Single Cyl.	Blowing Cyl.	Paris Exhibition and Cockerill Works.	May, 1900
800	Single Cyl.	Blowing Cyl. & Dynamo.	Hütte Aumetz Friede, Lothringen.	May, 1900
800	Single Cyl.	Dynamo.	Differdinger Hutte (Differdange).	Aug., 1900
800	Single Cyl.	Blowing Cyl.	Differdinger Hutte (Differdange).	Aug., 1900
800	Single Cyl.	Blowing Cyl. & Dynamo.	Differdinger Hutte (Differdange).	Aug., 1900
800	Single Cyl.	Dynamo.	Differdinger Hütte (Differdange).	Oct., 1900
800	Single Cyl.	Blowing Cyl.	Differdinger Hutte (Differdange).	Jan., 1901
800	Single Cyl.	Dynamo.	Rheinische Stahlwerke, Ruhrort, Germany.	Mar., 1901
800	Single Cyl.	Dynamo.	Differdinger Hütte (Differdange).	July, 1901
800	Single Cyl.	Blowing Cyl.	Cochrane & Co., Middlesbrough, England.	Nov., 1901
800	Single Cyl.	Blowing Cyl.	Roechlingsche Eisen- u. Stahlwerke, Volklingen.	Jan., 1902
800	Single Cyl.	Blowing Cyl.	Roechlingsche Eisen- u. Stahlwerke, Volklingen.	Jan., 1902
800	Single Cyl.	Blowing Cyl.	Société Ougrée Marihaye, Ougrée, Belgium.	Feb., 1902
[illegible]	Single Cyl.	Dynamo.	Société Ougrée Marihaye, Ougrée, Belgium.	Feb., 1902
800	Tandem.	Dynamo.	Société Anonyme John Cockerill, Seraing.	Mar., 1902
[illegible]	Single Cyl.	Dynamo.	Société Anonyme John Cockerill, Seraing.	Mar., 1902
800	Single Cyl.	Blowing Cyl.	Aachener Huttenverein, Esch.	May, 1902
800	Single Cyl.	Blowing Cyl.	Aachener Hüttenverein, Esch.	May, 1902
800	Tandem.	Dynamo.	Société Anonyme John Cockerill, Seraing.	June, 1902
800	Single Cyl.	Blowing Cyl.	Société Anonyme John Cockerill, Seraing.	June, 1902
125	Single Cyl.	Pumps.	De Wendel, Noyeuvre.	Apr., 1903
1600	Tandem.	Blowing Cyl.	Roechlingsche Eisen- u. Stahlwerke, Carlshütte.	Nov., 1902
1600	Single Cyl., Double-Acting.	Blowing Cyl.	Société Anonyme John Cockerill, Seraing.	Oct., 1903
125	Single Cyl.	Power.	Société Anonyme John Cockerill, Seraing.	Dec., 1903
[illegible]	Tandem, Double-Acting.	Dynamo.	Société Ougrée Marihaye, Ougrée.	Dec., 1903
800	Single Cyl.	Blowing Cyl.	Société Ougrée Marihaye, Ougrée.	Building.
1600	Single Cyl., Double-Acting.	Blowing Cyl.	Société Dnieprovienne du Nidi de la Russie à Kamenskoic.	Building
1600	Single Cyl., Double-Acting	Blowing Cyl.	Société Dnieprovienne du Nidi de la Russie à Kamenskoic.	Building.
1600	Tandem, Double-Acting.	Dynamo.	Société Anonyme John Cockerill, Seraing.	June, 1904
1600	Tandem.	Blowing Cyl.	Société de Vezin Aulnoye, Homecourt.	May, 1904
650	Tandem, Double-Acting.	Dynamo.	Usine de Bogoslowsk, Russie.	Apr., 1905
650	Tandem, Double-Acting.	Dynamo.	Usine de Bogoslowsk, Russie.	Apr., 1905
2000	Single Cyl., Double-Acting.	Blowing Cyl.	Société Minière d'Elba, Porto Ferrais.	Mar., 1905
1600	Tandem, Double-Acting.	Dynamo.	Société Anonyme John Cockerill, Seraing.	Mar., 1904
1600	Tandem, Double-Acting.	Dynamo.	Exposition Universelle de Liège, 1905.	Built May, 1905
650	Twin, Double-Acting.	Dynamo.	Exposition Universelle de Liège, 1905.	
1600	Single Cyl., Double-Acting.	Blowing Cyl.	Cie des Forges et Acieries de la Marine, de et à Homecourt.	Building.
1600	Single Cyl., Double-Acting.	Blowing Cyl.	Cie des Forges et Acieries de la Marine, de et à Homecourt.	Building.
1600	Tandem, Double-Acting.	Dynamo.	Société Ougrée Marihaye, Ougrée.	Building.
1600	Tandem, Double-Acting.	Dynamo.	Société de la Providence, Marchienne-au-Pont.	Building.
700	Single Cyl., Double-Acting.	Blowing Cyl.	Société de la Providence, Marchienne-au-Pont.	Building.
700	Single Cyl., Double-Acting.	Blowing Cyl.	Société Métal, Donetz Juriewka, Juriewka.	Building.
700	Single Cyl., Double-Acting.	Blowing Cyl.	Société Métal, Donetz Juriewka, Juriewka.	Building.
700	Single Cyl., Double-Acting.	Blowing Cyl.	Société des Acieries de France, Isbergues.	Building.
1600	Single Cyl., Double-Acting.	Blowing Cyl.	Société Anonyme John Cockerill, Seraing.	Building.
125	Single Cyl., Double-Acting.	Pumping.	Société Ougrée Marihaye, Ougrée.	July, 1903
1600	Single Cyl., Double-Acting.	Blowing Cyl.	Aachener Hütten Verein.	Building.
1600	Single Cyl., Double-Acting.	Blowing Cyl.	Aachener Hütten Verein.	Building.
1600	Single Cyl., Double-Acting.	Blowing Cyl.	Aachener Hütten Verein.	Building.

dertaken the construction of blast-furnace gas-engines. This company, which has acquired licenses for the construction of the Körting two-cycle double-acting engine, exhibited an engine of this class at the Liège Exhibition. It is installing two engines of this type at the blast-furnaces of Grivegnée, but the results have not, as yet, been made public.

Appendix I.

The progress made in the design and construction of blast-furnace engines can best be seen by the list of engines given in Tables III. and IV., which, up to the present time, have been built, or are building, by the Cockerill Company.

Table IV.—*Additional Orders for "Cockerill" Type Gas-Engines.*

I. H. P.	Type.	Driving.	Building for
1600	Single Cyl., Double-Acting.	Blowing Cyl.	Compagnie des Forges & Acieries de la Marine & à Homecourt
1600	Single Cyl., Double-Acting.	Blowing Cyl.	Compagnie des Forges & Acieries de la Marine & à Homecourt.
700	Single Cyl., Double-Acting.	Blowing Cyl.	La Société des Forges de la Providence, Marchienne-au-Pont
700	Single Cyl., Double-Acting.	Blowing Cyl.	La Société des Forges de la Providence, Marchienne-au Pont.
1800	Tandem, Double-Acting.	Dynamo.	La Société des Forges de la Providence, Marchienne-au-Pont.
1800	Tandem, Double-Acting.	Dynamo.	La Société des Forges de la Providence, Marchienne-au-Pont.
850	Single Cyl., Double-Acting.	Blowing Cyl.	Société Ougrée Marihaye, Ougrée.
300	Vertical Twin, Double-Acting.	Dynamo.	Société Usines Maltzoff, St. Petersburg.
300	Vertical Twin, Double-Acting.	Dynamo.	Société Usines Maltzoff, St. Petersburg.

Appendix II.

In addition to the engines enumerated in Appendix I., the following engines have been built, or are building, at the works enumerated, under licenses from the Cockerill Company:

Table V.—*Additional "Cockerill" Engines being built under license from the Cockerill Company.*

Name of Firm.	No. of Engines.	Aggregate I. H. P.
Richardsons, Westgarth & Co., Ltd	24	19,650
Schneider & Co.	15	7,880
Société Alsacienne	10	8,250
Société Française des Constructions Mécaniques	8	7,200
The Markische Maschinenbauanstalt in Wetter a. d. Ruhr	13	15,780
Breitfeld, Danek & Co., Karolinenthal, Prague	10	6,660
Elsässische Maschinen Fabrik, Mülhausen	33	38,130

The Application of Large Gas-Engines in the German Iron and Steel Industries.*

BY K. REINHARDT, DORTMUND, GERMANY.

(London Meeting, July, 1906.)

THE idea of burning blast-furnace gases directly in gas-engines, instead of under steam-boilers, as had previously been done, was first put into practice barely ten years ago, almost simultaneously in Great Britain, Germany and Belgium.

The pioneers of this movement made their first experiments with small engines. After these experiments had given satisfactory results, and it had been shown that the thermal value of the poor blast-furnace gases, in spite of defective scrubbing, could with safety be directly transformed into mechanical work in the gas-engine, there very soon arose a considerable demand for gas-engines of large power, similar to the large steam-engines employed in metallurgical works.

In face of these sudden requirements, the gas-engine builders were placed in a difficult position; for up to that time the gas-motor was considered as suitable for small engines only, and it was generally believed that the limit of size had been attained with 100 to 150 effective h.p. in a single cylinder.

Manufacturers of gas-motors, however, did not underrate the advantages offered by this new field for their manufactures, and it so happened that in Germany the Berlin-Anhaltische Maschinenbau-Gesellschaft, of Dessau, was the first to undertake the construction of a 600-h.p. two-cycle gas-engine, with two cylinders of the Oechelhäuser-Junkers type, for the Hoerder Mining & Smelting Company. The engine was started in 1898, and, with a few improvements and alterations, is still working satisfactorily. This engine worked with astonishingly high efficiency, considering the period. The builders were therefore

* Presented at the Joint Meeting of the Iron and Steel Institute and the American Institute of Mining Engineers, London, July, 1906, and here published under a mutual agreement between the Councils of the two Institutes.

in a position to prove in Germany that the gas-motor, even as a large engine, was suitable for the utilization of blast-furnace gases. The results obtained by the Hoerder Smelting Company, combined with the fact that the use of blast-furnace gases in gas-engines is much less dangerous than the burning of the same under steam-boilers, and is at the same time from three to four times as efficient,[1] encouraged other iron-works—principally the Friedenshütte and the Differdingen iron-works—similarly to introduce gas-engines; and during the last few years the example of these iron-works has been followed by a number of collieries, with a view to the more perfect utilization of the gases from their coke-ovens.

Even though, as was only natural, all these early gas-engines showed certain defects of design and insufficient purification of the gas, nevertheless, from the results obtained, and especially from those of the large installations of gas-engines supplied by the Cockerill Company, of Seraing, to Differdingen, it could be concluded that, with a small reserve, the power supply of an iron-works and, to a certain extent, of a rolling-mill, could be assured with gas-engines as the motive power, without serious disturbance of the existing arrangements. This can no longer be doubted when the improvements in their designs made by engineers engaged in the manufacture of gas-engines, as shown by the Oechelhäuser two-cycle motor, the Körting double-acting two-cycle motor, and the Deutz double-acting four-cycle motor, are considered. The Maschinenbau-Gesellschaft of Nürnberg, the Cockerill Society and other manufacturers have also entered the field. It would not otherwise have been possible for the application of blast-furnace gas to have become so widely spread in so short a space of time.

The present paper may be regarded as a continuation of those read before the Society of German Ironmasters by Dr. Lürmann,[2] Professor Meyer,[3] and myself.[4]

The object of the present paper is to review:

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[1] *Stahl und Eisen*, vol. xix., p. 484 (1899).

[2] *Ibid.*, vol. xviii., pp. 247 to 267 (1898); vol. xix., pp. 473 to 489 (1899); vol. xxi., pp. 433 to 459, 489 to 508 (1901).

[3] *Ibid.*, vol. xix., pp. 517 to 532 (1899); vol. xxv., pp. 67 to 72, 132 to 144 (1905).

[4] *Ibid.*, vol. xxii., pp. 1157 to 1182, 1352 to 1357 (1902).

(I.) The extent of the application of gas-engines in iron-works and collieries in Germany;

(II.) The working results, including the influence of purification on the gases; and

(III.) Present practice in the design of large gas-engines in Germany.

In order to arrive at conclusions as correct and complete as possible, the iron-works and collieries possessing gas-engines were invited to answer a series of questions, and the manufacturers of gas-engines to supply detailed drawings. And in taking this opportunity of thanking them, I wish to say that I received from all the iron-works and collieries complete and descriptive replies, which I was authorized to make use of, to my questions; the information received from the manufacturers being sufficient to enable the various types to be compared.

The questions, which were circulated through the kindness of the Society of German Ironmasters in February of this year, were as follows:

How many gas-engines have you at work, in course of erection, or on order? What type do they belong to? What size are they? What are they used to drive? What proportion of horse-power of the above are for continuous working, and how many are kept as reserve?

With what kind of gas are the engines worked—blast-furnace gas, coke-oven gas, or a mixture of both?

Is a gas-producer plant provided as a reserve?

How are the gases purified and cooled?

What power and how much water are employed for the purification of the gases consumed by the engine?

What proportion does this power bear to the power obtained from the purified gases?

What is the cost of purifying per cu. m. of gas?

What apparatus is employed to dry the gases after purification?

What is the percentage of dust contained in the gases before and after purification?

Are pressure-regulators arranged in the gas-main to the engines, and, if so, are they provided for each engine or for the whole installation?

What is the capacity of the pressure-regulator?

What is the pressure of the gases before they reach the engine, and within what limits does it vary?

At what temperature and with what percentage of moisture in grams per cu. m. is the gas supplied to the engines?

After what period of working is a complete cleaning of the internal parts of the whole apparatus undertaken, and how long does this take?

Which parts require cleaning first of all, or more frequently than the rest, and how long does the cleaning of these parts take?

Do stoppages and troubles occur, and what are the causes of these: (*a*) broken springs? (*b*) hanging of the valves? (*c*) failing to start? (*d*) failure of ignition?

Have important parts of the machine already become defective, and how long after the first starting of the engine? What do you consider the causes: (*a*) cylinders? (*b*) cylinder-covers? (*c*) pistons? (*d*) valve-boxes? (*e*) piston-rods?

How much water is used for cooling per horse-power and per hour: (*a*) for the cylinders? (*b*) for the pistons and rods?

At what pressure is the water used for cooling the pistons?

How much fresh oil is used per effective horse-power and per hour in the case of: (*a*) cylinder oil? (*b*) machine oil?

Has the consumption of gas by the engine been determined? If so, by what method, and what was it?

What sizes of units are, in your opinion, the most suitable for blowing-engines and for driving dynamos?

Can alternating-current dynamos driven by gas-engines be coupled in parallel without difficulty in your works?

A similar series of questions dealing with the special conditions at collieries was also sent out.

I. The Extent of the Application of Gas-Engines in Iron-Works and Collieries in Germany.

I received answers to the questions in the beginning of March, 1906.

From the inquiry it was ascertained that of the 49 German smelting-works questioned, 32 already had gas-engines at work and 9 had ordered such engines.

		Horse-Power.
There were at work—	203 engines having a total effective power of about	184,000
In course of erection, erected, and on order,	146 engines having a total effective power of about	201,000
Together,	349 engines having a total effective power of about	385,000
	Of these engines there were:	
	64 having a power of about of older construction (single-acting four-cycle motors)	34,000
	88 engines having a power of about two-cycle motors, and	91,000
	197 engines having a power of about double-acting four-cycle motors.	260,000
	For driving blowing-engines there will be at work:	
	15 old form single-acting four-cycle motors of about	8,200
	44 two-cycle motors of about	50,100
	77 double-acting four-cycle motors of about	103,000
Together,	136 engines. Of about	161,300
	For driving dynamos there will be at work:	
	48 older model single-acting four-cycle motors of about	25,600
	41 two-cycle motors of about	35,700
	110 double-acting four-cycle motors of about	144,800
Together,	199 engines. Of about	206,100
	For driving rolling-mills there will be at work:	
	0 older model single-acting four-cycle motors,	0
	3 two-cycle motors of about	5,200
	7 double-acting four-cycle motors of about	10,900
Together,	10 engines. Of about	16,100
	For other purposes:	
	4 engines of about	1,500

There were ordered by German iron-works and collieries from March 1 up to July 1, 1906:

		Horse-Power.
	7 two-cycle motors of about	7,800
	24 double-acting four-cycle motors of about	28,350
Together,	31 engines of about	36,150
	Of these engines there will be working, for driving blowing-engines:	
	7 two-cycle motors of about	7,800
	7 double-acting four-cycle motors of about	9,400
Together,	14 engines of about	17,200

		Horse-Power.
	For driving dynamos:	
	0 two-cycle motors of about	0
	17 double-acting four-cycle motors of about .	18,950
Together, . . .	17 engines of about	18,950
	For driving rolling-mills:	
	0 engines of about	0
	For other purposes:	
	0 engines of about	0

The largest aggregate power of gas-engines at a single works amounts to 35,000 effective h.p.; 16 works possess over 10,000 h.p., and 27 works possess over 5,000 h.p. in actual working.

In most iron-works the whole of the gas-engines work continuously without any reserve; a few have up to 40-per cent. reserve of gas-engines, and a few have a similar reserve of older types of steam-engines or of steam-turbines.

Nearly all engines in iron-works, naturally, work with blast-furnace gases. Two plants use only coke-oven gases, three use blast-furnace gas and coke-oven gas separately, and one plant uses the two gases mixed. Further, the Mansfeld Company utilizes the waste gases from the copper-smelting furnaces for driving gas-engines. Producers employing coke as fuel are kept as a reserve at seven works. They are really only of use in case of a strike, to assure the working of the most necessary part of the plant.

The application of gas-engines in collieries is much less important. This is due to the fact that the heat given off by the older type of coke-oven can only be utilized under steam-boilers, and consequently, for these older plants, steam-boilers are inevitable in collieries. Only the excess gas produced in the coke-ovens is available, so that steam-engines and gas-engines will always be found in conjunction, and indeed in larger proportion as regards steam-engines than is the case in iron-works. In the new regenerative coke-ovens the waste heat is utilized for pre-heating the oven itself, whereby there is an economy in gas, and a greater excess of gas is available for driving gas-motors. The irregular production of gas can, however, only be considered available or free from drawbacks for motor-driving when at least 60 coke-ovens are in operation. To this must be added, that the production of gas in coke-ovens is much more irregular than in blast-furnaces.

Perhaps in the near future, in addition to the excess coke-oven gases used for driving motors in collieries, producer-gas, which will be obtained in the ring-producer patented by Bergrat Jahns,[5] may also be used. The chief object of this producer is the utilization of the waste-heaps or culm-banks, and the production of gas as far as possible free from tar. The gas from the producer mentioned is naturally also suitable for the driving of gas-engines, which is demonstrated by the gas-engine plant at the Von der Heydt mine.

The same object is effected by the Turk and other producers; but, so far as I am at present aware, the utilization of the waste-heaps and of low-grade coal in gas-producers is only just being introduced, so that, for gas-engines in collieries, at present only coke-oven gas need be taken into consideration. So far as I am aware, in the beginning of March of this year, 16 collieries possessed 35 gas-engines at work, in course of erection, or on order.

The aggregate power of all these engines was 30,300 effective h.p. Of these, 24 engines were already working at 15,600 effective h.p., nearly all for the production of electricity.

The introduction of large gas-engines in collieries was subsequent to their introduction into iron-works, and, therefore, only engines of comparatively modern construction are to be found. Exception must, however, be made in respect to the smaller motors, which were early employed in plants for the recovery of the by-products in coke-oven gas.

II. Practical Experience Obtained in Working.

From what has been learned up to the present time it is clear that a thorough purification and drying of the gas is undoubtedly the principal factor in assuring a continuous and undisturbed working of gas-engines. German gas-engine manufacturers have from the very first considered the cleaning of the gas an essential condition, while, on the other hand, the Cockerill Company considered it unnecessary.

As a matter of fact, at many places Cockerill engines were working satisfactorily without any cleaning whatever, while at

[5] *Zeitschrift des Vereines deutscher Ingenieure*, vol. xlviii., p. 311 (1904).

other works this practice resulted in very disagreeable experiences; in one case, owing to the excessive wear of the working-surfaces of the cylinder, and another time owing to premature ignition caused by the formation of a crust, chiefly on the piston-ends, favored probably by excessive lubrication. If at one works the engines gave satisfaction without the gas being cleaned, and at another works similar engines were unsatisfactory, this only proves that the gas contains different percentages of dust at various works as it issues from the furnace-throat, and that it may become partly cleaned in the gas-main. It shows further that the same quantity of dust does not everywhere have the same effect, since it may be composed, in some works, of soft substances which do not so quickly cause excessive wear of the working-surfaces.

The design of the older Cockerill engines was, moreover, as regards the inlet-valves, not very sensitive to the effect of dust, the units most frequently constructed being 600 h.p. in one single-acting cylinder, and in consequence the sections of the gas-passages before the valve and of the inlet-valve itself were of rather large dimensions, and parts likely to be injured by the dust were not present with the system of governing then employed.

The methods of governing and of mixing the gases in newer constructions, in which stringent specifications for smaller variations of speed are laid down, are much more sensitive to the presence of dust, owing to their being combined with springs as delicate as possible, in order to keep the resistance of the governor and back-pressure upon it as low as possible.

If the spindles or regulating slide-valves are covered with a coating of dust, for instance, the springs are no longer sufficiently powerful to move these parts at all, or at the right moment, and in consequence disturbances in working result. This also occurs if dust is deposited on the valves or slides, the positions of which are regulated by the governor according to the load on the engine. The valves and throttle-valves (manipulated by hand) of the gas-main leading to the engine are also very sensitive to dust. The dust deposits on them very readily, and renders them difficult to move, and the areas at these places are for the time being unduly restricted, so that the engine does not receive sufficient gas to maintain

its normal power. In all the above cases, in addition to the percentage of dust, the percentage of water contained in the gas when admitted to the engine also exercises an injurious effect.

It is easily understood that moist dust adheres with greater facility to the surfaces with which it comes in contact than dry dust, the greater part of which passes through the engine without being deposited.

Great trouble is experienced with moist and dusty gas when the engine does not run continuously, but stops working on Sundays, for instance. It may then happen that the deposit of wet dust, which, while the engine is continuously working, does not offer very great resistance to the motion of the valve-gear, dries to a hard crust while the engine is not running, and causes these moving parts to become jammed, rendering the starting of the engine impossible.

The circumstances mentioned above are the result of the gas not being sufficiently purified or dried, as well as of the greater consumption of oil necessitated, and the consequent increase of dirt inside the motor, and, as a matter of fact, are the cause of most of the troubles experienced in working. For this reason, in all new plants, great importance is attached to the effective cleaning of the gas.

Before the introduction of blast-furnace gas-engines the washing of blast-furnace gas was considered necessary or advantageous for blast-heating and steam-raising purposes; the presence of dust diminished the efficiency of the combustion and of the heat-transmission, and rendered frequent cleaning of the hot-blast stoves necessary, although cleaning the gas for the above purposes did not have to be carried out to the degree necessary for the working of motors.

The whole of the gas from the blast-furnace is now subjected to a certain amount of washing, determined by experience, while the gas destined for utilization in the motors undergoes still further purification.

For a standard type of purifying plant for blast-furnace gas, the following may be observed:

The gases on leaving the blast-furnace are led through a series of so-called dry purifiers, and thence through long pipe-lines into the coolers or scrubbers, and from these into the

so-called centrifugal purifiers (Theisen apparatus or fans with water-spray). After leaving the above plant the purification of the gas should be complete, so that before being admitted into the engine the gas has only to be dried in filters or in capacious tanks.

In several plants, by drying or by passing through a long main to the engine, a further noteworthy purification of the gas takes place.

With regard to the construction and manner of working of the various apparatus, the following remarks may be made:

The dry purifiers consist generally of a combination of cylin-

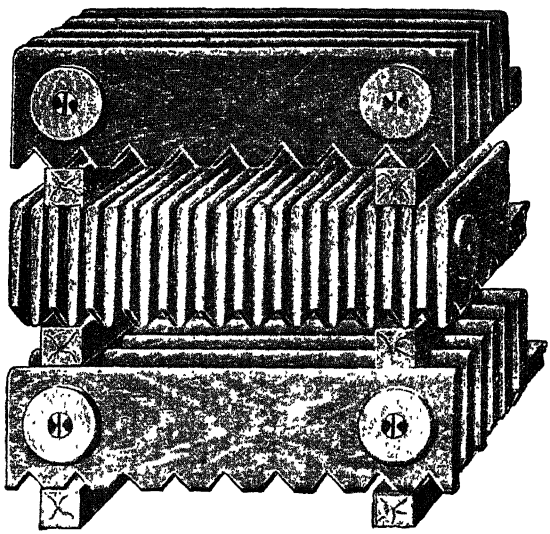

FIG. 1.—THE ZSCHOCKE PATENT SCRUBBER.

drical vessels, in which the gas is led downwards with a rapid motion and upwards with a slow motion. During this movement, and especially during the change of direction of the stream of gas, the coarsest particles of dust are separated. The pipes leading from the above should be made as long as possible, with as large a section and as many sudden changes of direction as possible, in order that the gas may be further freed from coarse particles of dust.

The coolers or scrubbers are vessels in which the gas flows from the bottom to the top and the water from the top to the bottom. The water must be finely sprayed in order to moisten

the dust, and thereby increase its weight and cause it to settle to the bottom. At the same time the gas is cooled in the scrubbers, in which the water-vapors are condensed and the dust is deposited.

The vessels are either empty, in which case the water is finely divided by spraying-nozzles, or the interior is arranged with sieves, wire-netting, coke, or wooden trays. The best example of the latter form is the Zschocke scrubber (Fig. 1).

The interior of the Zschocke scrubber consists of a series of wooden trays, one above the other, intended to reduce the velocity of the falling water, and by reason of their special form to divide the water into fine streams, so that the large surface exposed may effect a satisfactory cooling of the gas. The precipitated dust is removed at the bottom of the scrubber.

In centrifugal purifiers the further separation of the dust is effected by centrifugal action on the wet dust. The application of this apparatus renders it possible to attain a satisfactory purification of blast-furnace gas. The first centrifugal purifier in Germany was the Theisen apparatus, patented by Mr. Theisen of Munich.

In Duedelingen it was subsequently discovered accidentally that an ordinary fan, with water injection, was also very efficacious for gas-cleaning.[6]

The Theisen apparatus (Figs. 2, 3, 4, 5), according to a description given by one of the makers, the Dingler Maschinenfabrik, of Zweibrücken, consists essentially of the following parts:

The suction-chamber, A.

The pressure-chamber, B.

The middle chamber, C.

The drum with shaft and bearings, D.

The grating, E.

Water enters at F tangentially to the casing of the middle chamber, C, and leaves the apparatus through the pipe, G.

The manner of working this apparatus may be described as follows:

After the gas has been cooled and charged with water-vapor, it is drawn in by the vanes, h, and the coarse dust is separated in the suction-chamber. Through the action of the fans at both ends of the drum, D, the gas is then drawn through the space

[6] *Stahl und Eisen*, vol. xxi., p. 447 (1901).

between the drum and the casing. The outer circumference of the drum (Figs. 2–5) is provided with a number of inclined spiral vanes, i, so that the gas has also to travel a long way in

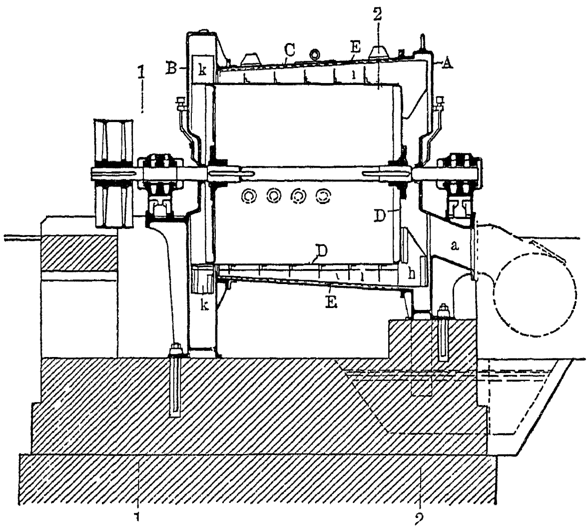

Fig. 2.—The Theisen Gas-Washer. (Cross-Section.)

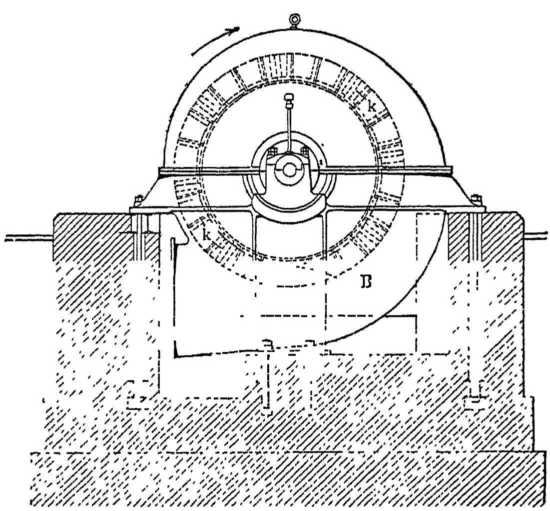

Fig. 3.—The Theisen Gas-Washer (Section on Line 1-1 of Fig. 2).

the form of a spiral. Hence, by injecting water at the same time through the pipes, F, a high degree of purification of the gas takes place, and the accompanying water-vapor is simulta-

neously condensed. The dust is then projected against the spiral meshes of the coarse grating, E, fixed to the interior surface of the casing. By centrifugal action, the water entering

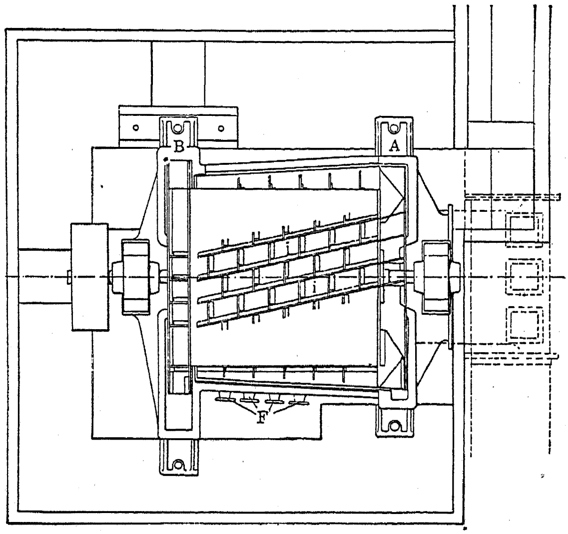

FIG. 4.—THE THEISEN GAS-WASHER. (Plan.)

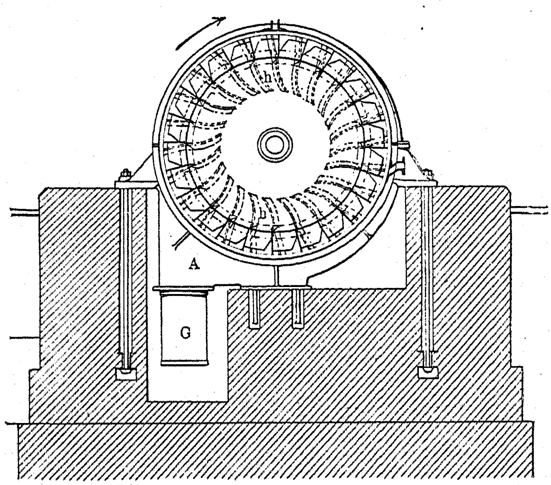

FIG. 5.—THE THEISEN GAS-WASHER (Section on Line 2–2 of Fig. 2).

in a tangential direction is at the same time distributed over the surface of the grating, which prevents it from becoming clogged and incrusted with the separated dust. In addition to this, the surface of the water is broken up, thus favoring cooling and

condensation. This washing of the gas absorbs carbonic acid and sulphurous gases.

The purified gas then enters the discharge-chamber, B, from which the water is thrown out by the vanes, k, and the gas is forced to the engines with a pressure of from 50 to 100 mm. of water. The washer reduces the dust from 3 or 4 g. per cu. m. of gas to 0.02 or 0.03 g., with a consumption of from 0.8 to 1.5 liters of water per cu. m. The washer is generally driven by a direct-coupled electro-motor, and the smaller sizes by belting, at a speed of from 300 to 450 rev. per min. The sizes generally used range from 6,000 to 33,000 cu. m. per hr., and the power required from 50 to 150 effective horse-power.

Theisen imputes the useful action, during purification, in his washer to the steam present in the blast-furnace gas and to that formed by contact with the injected water, and on this account recommends his apparatus to be placed, not behind the scrubber, but without such apparatus, by introducing simple gas pre-moisteners immediately behind the dry purifier, in order that the gas may be as hot as possible at the entrance into the apparatus. On the other hand, Professor Osann,[7] in an exhaustive investigation of the purification of blast-furnace gases, chiefly by the action of cooling-surfaces for the water-vapors and the deposition of dust, considers it preferable to clean and cool the gases previous to their being introduced into the Theisen washer, so that the latter has only to remove the finer particles of dust which are otherwise difficult to separate. He hopes by this arrangement to effect a saving of power.

The fans employed for the purification of the gases, as constructed by R. W. Dinnendahl at Steele (Fig. 6), only differ from ordinary air fans in the construction of the vanes and bearings, which are of a much heavier construction, to cope with the injection of water and the higher temperature of the gas. They are provided with a water-inlet at the suction-opening, and with an arrangement, as in disintegrators, for pulverizing the water, so that a sort of water-curtain is formed through which the dust has to pass. The cohering particles of dust and water are separated by centrifugal action through which these particles are thrown against the inner circumfer-

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[7] *Stahl und Eisen*, vol. xxii., p. 153 (1902).

ence of the fan-casing. The under portion of the fan-casing opens into a tank, A, from which the separated slimes flow away and the purified gas escapes at the top outlet. The method of purification resembles that of the Theisen apparatus, except that in the former the passage for the opposing action of the gas and water is not so long.

The usual sizes of gas-cleaning fans, according to Dinnendahl, are from 15,000 to 70,000 cu. m. of gas per hr., requiring from 40 to 110 h.p. The circumferential velocity of the im-

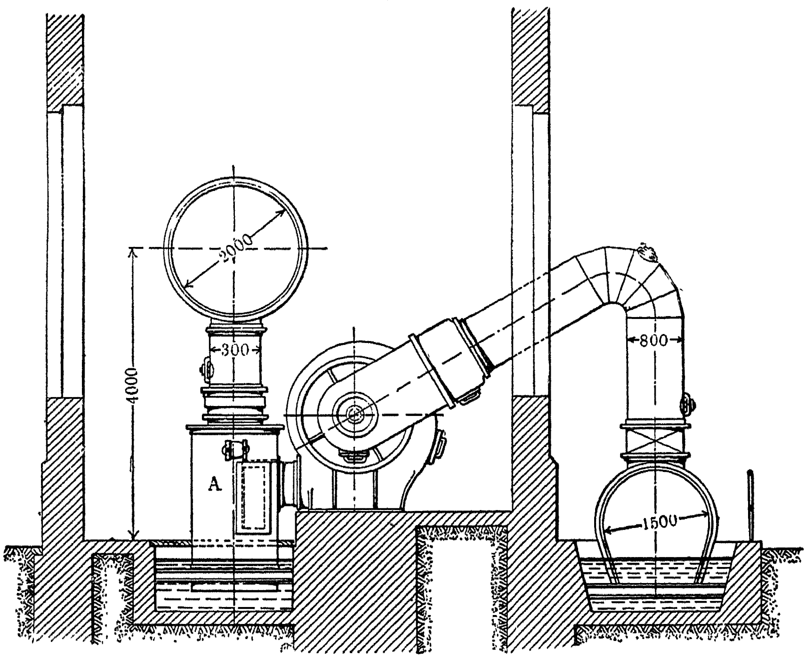

FIG. 6.—ARRANGEMENT OF FANS BY R. W. DINNENDAHL, STEELE.

pellers is up to 56 m. per second, with a diameter of from 1.1 to 1.75 m. For 1 cu. m. of gas from 1.5 to 2 liters of water are required, and the dust is reduced from 3 g. to 0.2 g.; as a rule, the percentage of dust is reduced to one-tenth of the percentage before washing.

When two or more fans are arranged parallel to one another for the purification of large quantities of gas, it is often difficult to obtain outputs equal in quantity and quality. It is therefore advisable to provide regulating-dampers behind the fans, and,

above all, to make the mains, both before and after the branches to the fans, of large diameter, so that they can at the same time act as air-vessels. As a certain preventive of the above difficulties, which are often of a very annoying character, I can offer only one suggestion—namely, to drive the fans and the electro-motors alike with the same speed in such a manner that their axes could be connected with friction-couplings, so that the fans produce equal differences of pressure.

Of the other purifying apparatus employed, only the Bian cooler may be mentioned.[8] This consists of a horizontal shaft turning within a cylindrical casing and carrying a number of disks of wire-netting. The lower halves of the disks dip into water, and the gas passes through the meshes of the upper halves.

The purification of the gas is continued in centrifugal apparatus until the desired degree of cleanliness is attained, after which it has only to be dried. This is effected by forcing the gas through a series of layers of wood-fiber or wool in large cylindrical casings, to which it yields its water. Naturally, the resistance caused in passing through the layers of wool requires a large expenditure of power, and the renewal of the wet wool, together with the cost of attendance, necessitates the installation of a spare drier. Large vessels containing various materials through which the stream of gas is forced, with frequent changes of direction, are employed for the separation of the water, and these vessels are further aided by long pipes with frequent changes of direction. If a large gas-holder is erected between the cleaning-plant and the engines, in addition to its quality as a pressure-regulator, it does excellent service in the separation of water, and renders the previous drying of the gas and the expenditure for attendance on the plant and power superfluous.

It must here be mentioned that in several iron-works it was not found possible to reduce the percentage of moisture in the gas arriving at the engine to the point of saturation at the corresponding temperature of gas. In such cases, after the supply of water to the scrubbers had been cut off, so that they were only employed as dry-coolers or purifiers, the gas was not so perfectly cleaned, but was drier, and worked with less harmful results in the gas-motors than before.

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[8] *Stahl und Eisen*, vol. xxvi., pp. 32, 465 (1906).

A few remarks concerning the purification of coke-oven gas for utilization in gas-engines must still be added.

The gas at disposal for this purpose has already been so far purified by the recovery of by-products that, as a rule, only the remains of tar, and also sulphur and cyanides, have to be removed. The tar-residues are removed in so-called tar-separators, which consist of high cylinders of boiler-plate in which a number of platforms or ledges are arranged alternately to the left and to the right, so that the gases pass through in a zigzag direction and the tar is deposited on the ledges. Other apparatus work in a similar manner, the main stream of gas being divided into a large number of smaller streams and by the resulting sudden alterations of direction, and also by impinging on the plate walls, the gas is freed from tar (Pelouze apparatus). Further, rotary cleaners are in use, which serve for the separation of ammonia, naphthalene, cyanide, and hydrogen sulphide, and according to the form of the rotating surface are arranged as hurdle, brush, or ball washers (patented by Zschocke[9]). The Theisen washer can also serve this purpose; but, as far as I am aware, it has not as yet been so employed. The inventor hoped to obtain good results, especially in the separation of tar.

The separation of sulphur and cyanide is, according to a paper of Professor Baum, best obtained by filters. The filtering-material employed consists of Laming composition, a mixture of bog iron-ore and wood-shavings. The composition, in layers of from 6 to 8 in. deep, is carried by plates or gratings; the gas passes through from two to four such layers, one after the other, and the iron combines with the sulphur to form iron sulphide, and with the cyanide to make iron cyanides (Berlin blue). The composition is from time to time taken out of the filter and exposed to the air, by which means the sulphur is oxidized and the composition regenerated and ready to be used again.

In passing through the filter, not only the sulphur, but also the tarry liquors, water, and heavy oils remain behind. For this reason, plants which do not require the removal of sulphur often employ filtering-apparatus, the Laming composition being replaced by sawdust or wood-fiber. Gasometers, which are frequently placed as near as possible to the engines, and, as in

[9] Baum, *Glückauf*, vol. xl., p. 457 (1904).

the case of blast-furnace gas, at the same time regulate the pressure, also serve to dry the gas.

With reference to the purification and its influence, the following may be seen from the answers to the questions: All smelting-works have centrifugal apparatus in use for removing the fine dust, and, indeed, about half of them have scrubbers or Bian coolers with fans, and the rest scrubbers with Theisen apparatus, Theisen apparatus alone, or fans alone. The respective merits of the various apparatus or processes cannot well be ascertained from the information received from the iron-works, as it is not easy to reduce the results to a common basis. The following results nevertheless are perhaps of interest:

The power expended in cleaning 1,000 cu. m. of gas per hr. varies mostly between 6 and 13 effective h.p. Accordingly, the power expended in cleaning varies from 1.8 to 4 per cent. of the power obtained by the purified gas.

The amount of water used for cleaning varies greatly. It requires on an average from 3 to 8 liters per cu. m. of gas, and is naturally dependent on the temperature of the water. Generally speaking, the water used with centrifugal apparatus alone is less than when it is employed in combination with scrubbers. Similarly, the cost of cleaning varies considerably, and includes interest and depreciation of the purifying plant (from 0.03 to 0.06 pfennig per cu. m.).

The percentage of dust in the gas after the dry purification is on an average from 4 to 6 g. per cu. m. In a few cases, however, it is only from 1 to 1.5 g. In most instances the gas for working the motors is reduced to a percentage of from 0.015 to 0.03 g. of dust per cu. m., in a few works even to from 0.004 to 0.005 g. per cu. meter.

All these remarks concerning the percentage of dust are to be judged from the point of view that the determination of the same at one and the same iron-works, if not absolutely correct, will always be proportionately exact; but that this latter will perhaps not always be the case with tests carried out by different iron-works. It would therefore be of importance to adopt a standard method for the determination of the percentages of dust and water, so that all results could be exactly compared.

If the purification effected by the Theisen apparatus is compared with that by fans, it will be found that, according to the

manufacturers, the Theisen apparatus cleans in the proportion of 140 : 1. Thus, for 1,000 cu. m. of gas cleaned per hr. there is required 5 effective h.p., and per cu. m. 1.15 liters of water, on an average.

With a fan the cleaning is on an average 10 : 1, the power required being 2.2 h.p., and the water used 1.75 liters.

In order to obtain a similar result, two to three fans would have to be placed one behind the other, which would require perhaps from 5 to 6 h.p. per 1,000 cu. m. of gas per hr., and a consumption of about 4 liters of water per cu. m. of gas.

From the information supplied by the iron-works only the total result can in most cases be reviewed; however, in a few cases the result of the cleaning by each apparatus is given, and from this I conclude that a single Theisen apparatus cleans better than a single fan, since with the former the proportion of cleaning is between 90 : 1 and 25 : 1, with about 6.5 effective h.p. per 1,000 cu. m. of gas, and with a fan the proportion is about 12 : 1 and the average effective h.p. 2.3. From two fans, one placed behind the other, a proportion of cleaning from 50 : 1 to 200 : 1 and power employed from 6.5 to 10 effective h.p. per 1,000 cu. m. per hr. has been attained. Without taking the consumption of water into consideration one Theisen apparatus is approximately equal to two fans.

With one exception, all iron-works possess apparatus for drying the gas as described above.

In no case does the gas contain any suspended water—that is, water above the quantity at the point of saturation at the corresponding temperature.

This temperature is in most cases the same as the temperature of the air, or only a few degrees higher. In a few cases the percentage of water is even lower than that corresponding to the point of saturation at the temperature of the gas, but this is only possible when the water used for cooling is at a very low temperature and the gas is cooled to below the temperature of the gas arriving at the end of the gas-main.

A further cooling of the gas would be of great utility, favoring the separation of water and purification, and thereby assuring the continual working of the gas-engines without disturbance.

The particulars which I have received from the collieries are

not so complete as those received from iron-works, owing to the collieries not yet having had so much experience.

Of 15 collieries which were questioned, two had no special plant for the purification of the gas, but only plant for the recovery of the by-products—four collieries have plant for the separation of sulphur and tar, six a similar plant for sulphur only, and three a plant for tar only. The power expended is only that necessary to overcome the resistance of the gas passing through the purifier, which is on an average about 0.25 per cent. of the power developed. The other working expenses consist only of the renewal of the filtering material, which amounts on an average to about 0.03 pfennig per cu. m., while the expenses of the purification-plant itself increase greatly with the sulphur in the gas.

Only traces of tar have to be removed by the purifier, but it is much more important to remove the sulphur, which attacks the cylinders, piston-rings, piston-rods, and stuffing-boxes.

In one case it is stated that the percentage of sulphur was reduced from 5 g. to 0.7 g. per cu. meter.

The heating-value of coke-oven gas varies from 2,500 to 4,600 calories per cu. meter.

The amount of gas available for gas-engines also varies extraordinarily, ranging from 3.25 to 50 per cent., according to the quality of the coal used, and above all according to the type of coke-oven.

In the replies to the questions addressed to iron-works, attention should be called to the fact that about one-half of the works place gas-holders between the purifying-plant and the motors. The capacity of the holders in proportion to the gas-consumption varies considerably. One iron-works places a gas-holder of smaller size, arranged as an equalizer of pressure, before each engine.

The pressure of the gas at the engines is on an average from 2 to 4 in., but in many plants it is 8 in. or over. The variations in the gas-pressure naturally depend on the number of gas-engines at work and of furnaces in blast, and on whether the blast-furnace tops are provided with a double seal or not. As a rule, it is recommended that the gas-pressure be maintained as regularly as possible, and not much above the pressure of the atmosphere (about equal to from 1.25 to 2.5 in. of

water). This can, of course, only be done by using a gas-holder, which, besides being an excellent separator for water, possesses the advantage of preventing a reduction of speed or even the stopping of the gas-engines when the supply of gas is suddenly interrupted for a short period, as may happen when only a small number of blast-furnaces are at work. Long gas-mains of large section also serve as a reserve, although not so effectively, and for a short period tend to equalize the pressure.

The intervals at which the engine or its several parts have to be cleaned vary greatly. From information received from iron-works, it may be concluded that with gas well cleaned (from 0.015 to 0.03 g. of dust per cu. m.), and at the same time well cooled and dried, the inlet-gear—that is, the parts before the cylinders of the engines—must be cleaned at intervals of two to three months, and a complete internal cleaning must be undertaken every six or eight months.

In a few plants using gas which is specially clean, the engines require less frequent cleaning. In others the inlet-gear, throttle-valves, and other similar parts require cleaning at periods of 14 days. At the same time, when the lubrication is not excessive, and even when the gas is not well cleaned, an internal cleaning of the engine every two to three months is sufficient.

The parts before the cylinder require for cleaning, on an average, from 6 to 20 hr., according to the size and build of the engine and the number of men employed, and the internal cleaning requires from two to eight days.

The quantity of water used for cooling the cylinders and pistons averages from 8.8 to 11 gal. per hr. and per effective h.p., of which from 2.2 to 2.6 gal. are for the pistons. The consumption of oil in most plants is reckoned at 1 to 1.25 g. per hr. per effective h.p. The consumption of gas has not yet been sufficiently tested to compare the various systems.

According to trials made at iron-works, the heat employed by the engines varies from 2,200 to 3,300 calories per hr. and per effective h.p. Most iron-works at present are not in a position to determine the consumption of gas in their engines, and content themselves with testing the exhaust-gases, and thereby determining the completeness of the combustion in the motor.

From the answers received from the collieries, engines using

coke-oven gas require cleaning after similar periods to those using blast-furnace gas.

Generally speaking, however, at present the collieries have not sufficient experience to answer this and other questions authoritatively. The traces of tar in coke-oven gas, which are difficult to remove and to burn, probably necessitate more frequent internal cleaning; and above all, the piston-rings, stuffing-boxes, oil-holes and other similar parts require greater attention.

III. The Present Design of Large Gas-Engines in Germany.

From the answers received from the iron-works and collieries to the questions, it may further be concluded that the old arrangement of the single-acting four-cycle motor, with one or more cylinders, has not, in recent years, been generally used, and that, on the other hand, double-acting four-cycle motors, mostly with tandem cylinders, are in keen competition with two-cycle motors. There is therefore little necessity to deal in this paper with the obsolete single-acting four-cycle motors, which have been replaced by newer designs, particularly as many of these engines were only considered by their designers as of temporary construction, owing to the heavy demand which suddenly arose for large gas-engines. To this category belong the older engines of the Gasmotorenfabrik Deutz, of the Société Cockerill, of the Körting Brothers, and of the Maschinenbau-Gesellschaft Nürnberg.

Most of the new types which are now prevailing have already been described.[10] In order to institute an accurate comparison with the types since designed, it will be necessary here to recapitulate as concisely as possible the chief features of modern gas-engines.

Before commencing, the fact may be recalled that the possibility of making gas-engines of larger powers depended on overcoming the numerous prejudices and misconceptions of many gas-engine manufacturers. These were chiefly with reference to the idea that it was impossible to construct a stuffing-box, a cooled piston-rod, and a cooled piston, which should be reliable in working.

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[10] *Stahl und Eisen*, vol. xxii., pp. 1157 to 1182, 1352 to 1357 (1902); vol. xxv., pp. 67 to 72, 132 to 144 (1905).

Once the possibility of constructing these parts to work in a reliable manner was proved by several engines built by Körting Brothers, John Cockerill, and the Maschinenbau-Gesellschaft Nürnberg, and after Körting Brothers had had a double-acting four-cycle engine running for a long time in their own works, and in the year 1902 made known their new double-acting two-cycle engine, a very rapid development took place. Many firms, which had previously built single-acting four-cycle engines, began to build double-acting closed cylinders and pistons working on both sides, so that in Germany at present, with the exception of the two-cycle Oechelhäuser motor, only the four-cycle Dingler engine has open cylinders.

To the best of my knowledge, at the present time 29 firms in Germany build large gas-engines. Of these, 21 firms build double-acting four-cycle engines, 5 firms build two-cycle engines, and 3 firms build both systems.

IV. General Remarks.

1. *Cylinder, Exhaust-Valve Chest.*

Until the year 1902 all German builders of gas-motors made their single-acting open four-cycle engines with cylinder ends, as used for small motors, the construction of which, owing to insufficient reliability, was the chief cause of the troubles encountered originally in the construction of large gas-engines.[11]

In the year 1902 the Gasmotorenfabrik Deutz[12] first introduced a design of cylinder (Fig. 7) which avoided the old cylinder ends, and was provided with an arrangement of lift-valves similar to a lift-valve steam-engine. The cylinder, which was provided with covers at both ends projecting within the cylinder in a similar manner to steam-engines, stood on a support in the center on which it could slide, and the outer mantle was, for part of its length, provided with an opening so that the semicircular turned support and semicircular cover at this place rendered the mantle water-tight. This manner of constructing the cylinder should avoid the danger of initial strains in casting, and reduce the stresses caused by heat in working,

[11] *Stahl und Eisen*, vol. xxii., pp. 1157 to 1182 (1902). [12] *Ibid.*, Plate 20.

and also render possible the complete removal of the core after casting, and an easy cleaning of the cooling-mantle.

In examining the various designs, it will be seen that nearly all cylinders of the newer types of four-cycle engines follow the same principles as to the arrangement of the lift-valves. Most makers, however, consider that a separation of the outer casing is unnecessary, and that, for safety, it is more to the purpose to increase the height of the flanges, and thus provide a larger space between the inner and outer mantles. It is easy

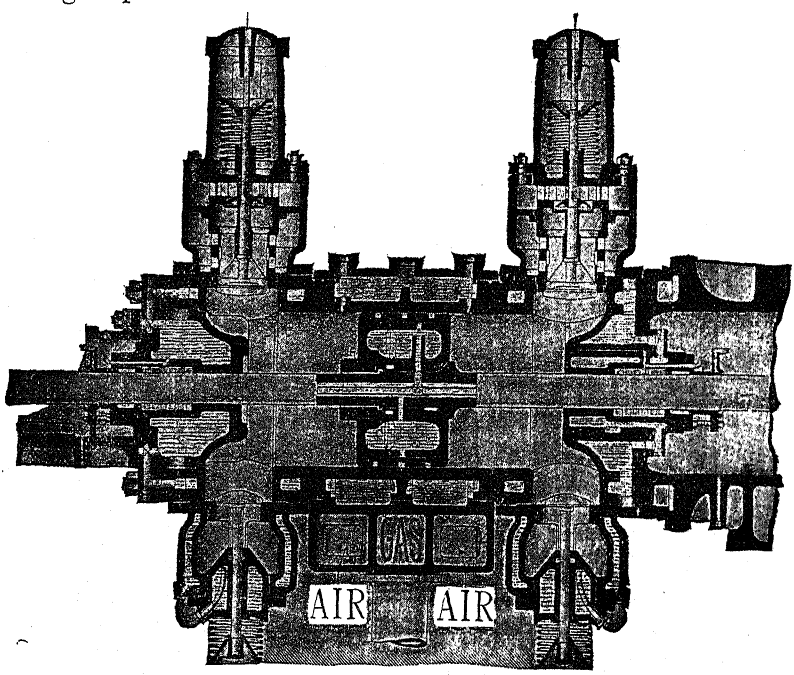

FIG. 7.—CYLINDER OF THE DOUBLE-ACTING FOUR-CYCLE ENGINE OF THE GASMOTORENFABRIK DEUTZ.

to understand that a larger flange is better fitted to resist the stresses due to the difference of temperature between the inner and outer covers, owing to the definite rates of expansion distributed over a greater length. On this account calculations can only be made with approximate accuracy, because the average temperatures, and, in particular, that of the inner mantle, are unknown.

Compared with the older design of cylinder-heads, with a strong flange (with which both the outer and inner casings of the head were cast together, and bolted to the cylinder proper,

and with the outer and inner casings rigidly connected by the branches of the valve-chambers), the newer types offer without doubt far greater security against breakages.

In the older form of cylinder-heads, the stresses caused by the heat were always much higher, at the portion of the head between the branches for the valves and the flange, than they are in any part of the newest form of cylinders. In the first mentioned, the inner surface of the rigidly connected parts was exposed through the whole of their length to the highest temperature at each explosion, while modern cylinders are much better in this respect.

With the latter this can be explained by the fact that, because the cylinder-covers project into the cylinder at both ends as far as the surface of the joint, the inner cylinder-walls are cooled both from without and from within (by the cooled walls of the cylinder-covers), and further, that the middle portion of the inner walls, or rather the working-surfaces of the cylinder, do not generally reach these high temperatures, and the whole working-surface is passed over by a cooled piston.

In this manner the average temperature of the inner wall remains considerably lower than was the case with the older cylinder-heads, and the design is also much more trustworthy. Many makers have ceased to cast the valve-chambers, which, with the inner cylinder, form one piece, together with the outer mantle, and thus increase the security of the construction.

This is the reason why, during the last few years, few instances of cracked cylinders have been heard of, and in exceptional cases where they have occurred, those who have investigated the subject are agreed that the cause of the breakage had nothing to do with the construction, but was to be attributed to the presence of water in the cylinder, to the mantle being badly washed in the foundry, and to the formation of deposits, and such like causes.

The exhaust valve-chambers also belong to those portions of a gas-engine which, like the cylinders, cylinder-covers, and pistons, are exposed to dangerous stresses caused by the fluctuating temperatures of their walls, so far as the latter form a single casting. Their design requires for this reason considerable care, and, above all, a symmetrical form.

The construction of the exhaust valve-chamber of the engine

of the Maschinenbau-Gesellschaft Nürnberg is an improvement well adapted for its purpose (Fig. 8). It fulfils at the same time the condition that the inner portion, which carries the valve-seating, can be withdrawn downwards with the valve, without disconnecting the valve-chamber, or its connection with the exhaust-pipe.

Similar constructions are made by the Gasmotorenfabrik Deutz, by Ehrhardt & Sehmer, and other makers.

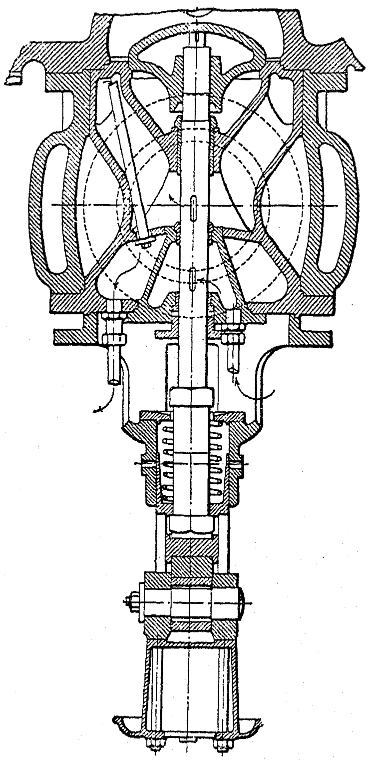

Fig. 8.—Exhaust Valve-Chamber of the Maschinenbau-Gesellschaft Nürnberg.

With several designs it is impossible to take out the valves without unbolting the connections of the whole chamber, with the exhaust-pipe, and removing the latter; for instance the engines made by the Elsässiche Maschinenbau-Gesellschaft, and the Märkische Maschinenbau-Anstalt (Plates IV. and V.). An arrangement of the exhaust valve-chambers differing somewhat from other constructions is that of Schüchtermann & Kremer (Fig. 37).

This chamber, situated at the side of the cylinder, as seen in the figure, is so constructed that the walls are entirely protected from stresses caused by the variations of temperature. The valve with its spindle can in this case be withdrawn upwards.

I wish to point out that, in order to silence the exhaust, a spray of water is employed with advantage in the exhaust-pipe, and the exhaust-vessel is made as large as possible. In such cases the exhaust-pipe must be provided with a drain-pipe of sufficient dimensions to allow the water to flow off freely, so that in case of negligence in the use of the water-spray—for instance, at the starting of the engine—no water can enter the cylinder through the exhaust-valve, and thus occasion its destruction.

Now that the causes of the less important breakages of cylinder-covers and of pistons have been found to be partly due to a faulty arrangement of ribs, these difficulties should in future be overcome by experienced makers. These accidents are generally discovered in time, by a slight leakage of water, and consequently the failing of the ignition and the falling-off of the power of the engine. It is to be hoped that most of the designs of cylinders, cylinder-covers, and pistons of the newer engines will prove to be permanently trustworthy.

2. *Valve-Gear.*

Included in the valve-gear of gas-engines, in addition to the mechanism which is arranged for the regular motion of the principal inlet- and outlet-valves on the cylinder, for the admission of the mixture, and for the exhaust of the burnt gases, respectively, are chiefly to be reckoned those parts which serve to regulate the speed, through the influence of the governor and the formation of the mixture.

The inlet-valve is always cooled by the fresh mixture admitted; it therefore requires no special cooling. It is, however, absolutely necessary that the hollow exhaust-valves should be water-cooled. Water is circulated through the hollow valve-spindle; at the same time it must be observed that the spindle must not be rendered water-tight by a stuffing-box, otherwise the valve easily hangs.

The valves are opened by an exterior mechanism driven by eccentrics or cams on a side shaft, which, in the case of four-cycle engines, runs at half the speed of the crank-shaft. The eccentric-rods in nearly all designs are combined with roller-levers.

Thus, in spite of the unavoidable acceleration of large masses of moving rods, and in spite of the pressure on the exhaust-valves when opened, the valves are lifted without shocks and the valve-gear works smoothly. The valves opening inwards are closed by springs.

The idea that cams are not suitable for operating lift-valve gearing is incorrect. A large number of gas-engines may be found working with valve-gearing controlled by cams, and the action of the latter is smooth and unobjectionable.

It is, of course, obvious that cams must be combined with

stronger springs than are necessary with eccentrics, because with the former, in addition to the valve, spindle, and roller-lever, the driving-rod of the gearing has also, as a rule, to be accelerated or moved by springs.

The strength of these springs, however, should not be greater, with regard to the acceleration of the mass, than necessary for other reasons—*i.e.*, when the engine is working by quantity governing (and therefore with constant mixture and variable compression); because, if it is running without load, the springs must be of such a strength that they will prevent the opening of the valve to the partial vacuum formed at the time of the suction-stroke (which amounts to about one-fourth atmosphere absolute).

By the arrangement of a double-curved cam, provided with a roller and a counter-roller, a constrained motion of the rod may be obtained both for opening and closing by the roller-levers without the aid of spring closure, and by a suitable design of the valve-gear, the constrainment can be extended just as well with cam as with eccentric, even to the valve by the introduction of buffer-springs. The springs have thereby only to endure a compression of a few millimeters. This arrangement, as a rule, is applied only to the exhaust-valve motion.

With eccentric valve-motions combined with roller-levers, the valve-rod and the active roller-lever always have to travel a long inactive distance, and therefore, as regards the admission-valve motion, usually require a long spring, having a compressive length equal to the travel of the valve.

In view of the satisfactory results of valve-motions, whether controlled by cams or by eccentrics, no general decision can be taken as to which design is the better for all cases.

The most important thing is to give the cam the correct form to assure smooth running; and makers of gas-engines, guided by experience, understand quite well how to do this, even though the method adopted is said not to be in accordance with the theory of the cam,

The valve-motions employed in large gas-engines for the purpose of governing and forming the mixture may be divided as follows:

(*a*) *The Quality Method of Governing, with constant total volume admission to the cylinder at each suction-stroke, and there-*

fore with constant compression, but supplying a variable mixture.—With this valve-gearing the composition of the mixture with a varying load is so varied through the actions of the governor, that (at a smaller power than the maximum) after opening the inlet-valve, air is first drawn into the cylinder and then at a certain position of the piston (depending on the momentary load on the engine and the position of the governor), the opening of the gas-valve commences, and the mixture continues to enter the cylinder until the end of the suction-stroke, when both the inlet- and the gas-valves close.

Therefore, with a smaller load, more air and less gas, but, with a greater load, less air and more gas, are drawn in. The compression remains constant, but the composition of the mixture during the suction-stroke is not only very variable with a varying load, but also with a constant load. Seeing that at first pure air alone is admitted, and that it is only afterwards that the gas is drawn in, the air has acquired an accelerated motion in the inlet-pipe, while the gas, which is allowed to enter gradually, starts from rest and has to accelerate; and in addition the gas has to flow through an opening, the area of which is continually altering during the period of the opening of the gas-valve. The composition of the mixture alters constantly, owing to the opposing influence of the air- and gas-pressures, and to the alterations of the area of the gas-inlet, which occur during the opening of the gas-valve. The old gear, which is now rarely used, for operating the gas-valve by inclined notches, is somewhat upon the same principle. With the latter gear, the gas-valve was brought to the closing position at the end of the suction-stroke, and then the trouble arose that, through throttling the gas, a still poorer mixture ensued. After the compression this was located exactly in the neighborhood of the ignition.

To get over this difficulty the gas-valve was not allowed to close till after the dead center was reached—*i.e.*, until after closing the inlet-valve; and so long as the inlet-valve remained open, the gas was not much throttled.

The principal disadvantages of the latter are: the weak mixture; with low loads, and with the engine running light, the irregular ignition caused by this, especially with varying gas-pressure, and the resulting uncertainty of governing, as well

as a proportionately large consumption of gas with smaller loads.

The slow consumption of the weak mixture often results in the fact that the exhausting-gases, and those gases which remain behind in the cylinder at the beginning of the suction, continue to burn and thereby ignite the incoming mixture afterwards, owing to which back-firing occurs in the suction-passage, which unfavorably influences the governing and the regular working of the engine, especially when running without load.

A valve-gear, based upon the same principles, is at the

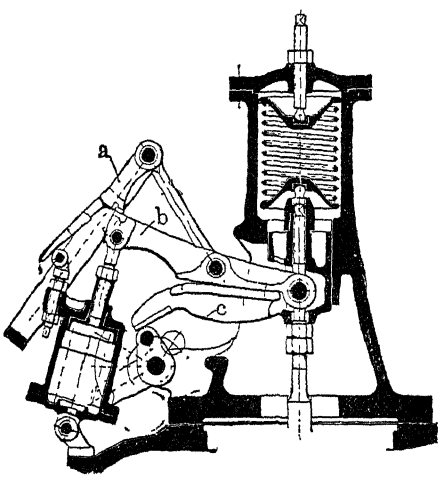

FIG. 9.—VALVE-REGULATION OF THE MASCHINENBAU-GESELLSCHAFT NÜRNBERG.

present time constructed by the Maschinenbau-Gesellschaft Nürnberg and their licensees (Figs. 9, 10).

The gas-valve is lifted by an active carrier, a, of a trip-gear jointed to an eccentric-rod and connected to an active roller-lever, b, which lies against a passive roller-lever, c, whose position is fixed by the governor. By this means the time of the opening of the gas-valve is dependent upon the position of the governor. When the carrier, a, is tripped, the gas-valve falls freely and closes at each charge, immediately with or shortly after the closing of the inlet-valve. This valve-gear—with the exception of the exterior mechanism, which is taken from

the modern steam-engine, and, moreover, is not free from a somewhat objectionable back-pressure on the governor—is in its action only an improvement on the inclined notch-gear.

The peculiarities of the quality method of governing described here induced the designer of the Nürnberg engine, Mr. Richter, to improve the valve-gear, with respect to the formation of the mixture, in the engines recently constructed under

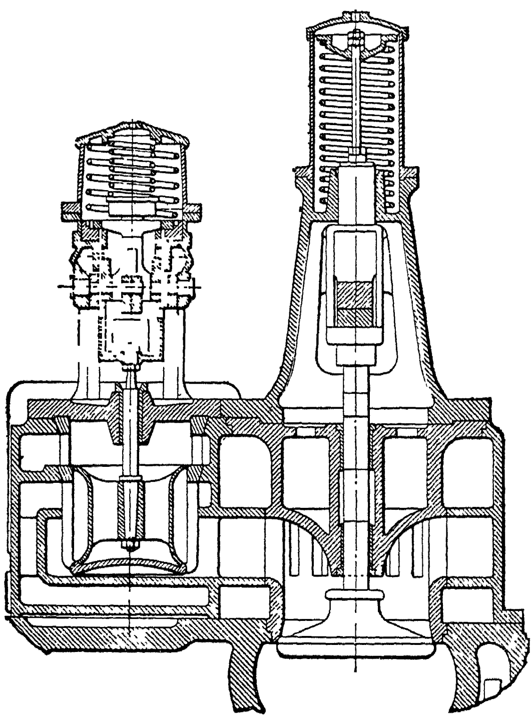

FIG. 10.—MIXING- AND INLET-VALVES OF THE MASCHINENBAU-GESELLSCHAFT NÜRNBERG.

his direction for the firm of Thyssen & Co., Mülheim-Ruhr. As shown by Fig. 11, a balanced double-seated valve is combined with a sliding-sleeve on the same spindle, which, when the gas-valve is shut, permits the admission of pure air to the inlet-valve through a slit which is always open. If the gas-valve is lifted, the sliding-sleeve increases the area of the air-passage regularly with the motion of the gas-valve. The object of this valve-gear is "to obtain as regular an acceleration and retarda-

tion of the air- and gas-columns as possible, without the partial vacuum, induced by an early cut-off, being too high. Further, as the gas-valve is double-seated, a good distribution of air and gas is obtained, and at the same time the acceleration of the air-column is utilized to accelerate that of the gas-column."

It is necessary, first, to ascertain from experience whether

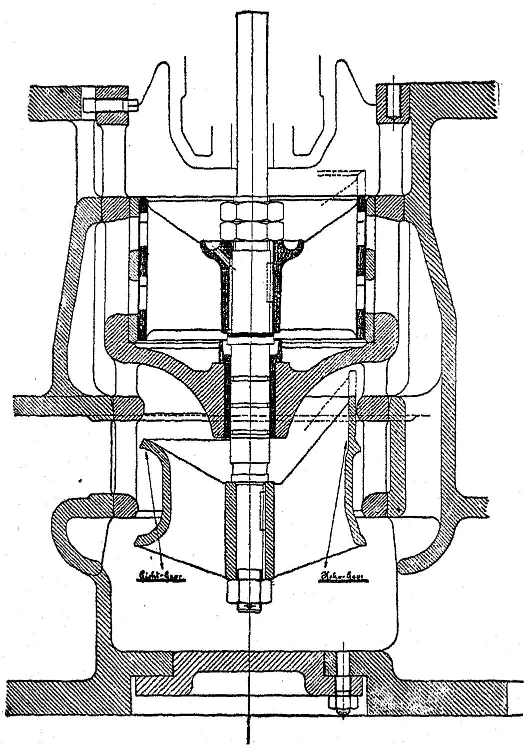

FIG. 11.—MIXING-APPARATUS OF THYSSEN & CO., MÜLHEIM A. D. RUHR.

this formation is really an advantage, and whether this is not out-balanced by the higher vacuum produced in the cylinder during the suction-stroke.

Should this arrangement of mixing-valve, as opposed to the ordinary quality governing, not result in the anticipated advantage of a more regular mixture with constant pressure, I recommend that the air-slide be made to work in an inverted man-

ner—*i.e.*, in such a way that it opens to its farthest extent when the gas-valve is shut, so that, after the gas-valve has begun to lift, it reduces to some extent the area of the opening for the admission of air. By this means, also, the unfavorable influence of the air-column first accelerated would be lessened.

The disadvantages of so-called quality governing are naturally less considerable in engines which work mostly on a load which varies little from the normal load, such as for the driving of blowing-engines and pumps. But for even driving of these latter I consider the following method of quantity governing preferable to quality governing.

(*b*) *Quantity Method of Governing, with varying cut-off, and therefore varying compression, but with constant mixture.*—In this method of governing it does not happen that, after the inlet-valve is opened, first pure air and then a continually varying mixture flows in, but from the very beginning of the stroke gas and air are admitted, and always in the same proportion, so that the condition of constant mixture is fulfilled—*i.e.*, if the diffusion of this with the residual gases is not taken into consideration. It is clear that this valve-gear must give a more regular mixture at the normal power than the quality method of governing. For lower loads the amount of the constant mixture is diminished by the action of the governor, either by throttling throughout the whole length of the suction-stroke, as in the valve-gear of the Gasmotorenfabrik Deutz (Figs. 24, 25), or by closing earlier by a cut-off arrangement (either a valve or a slide), which enables the air and also gas to be admitted in the desired proportions from the beginning of the suction-stroke. This last arrangement of quantity governing requires a special drive for this regulating-valve from the valve-gear shaft, but to compensate for this the negative work with a small load is less during the period of suction than with throttling.

This manner of governing, according to Professor Meyer,[13] gives, even with the engine running unloaded, an almost perfect and regular combustion, from which follows the possibility of obtaining effective governing when the engine is running almost without any load. The consumption of gas with small loads is also more economical than with the quality method of

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[13] *Stahl und Eisen*, vol. xxv., pp. 132 to 144 (1905).

governing, although, as compared to the consumption with larger loads, it increases, because the compression is reduced.

The advantages of this method of governing appear to be acknowledged by most of the older makers of gas-engines, with the exception of the Maschinenbau-Gesellschaft Nürnberg and their licensees. It is employed by Deutz, Cockerill, Körting, Elsässische Maschinenbau Actiengesellschaft, Ehrhardt & Sehmer, and others.

As regards the disadvantages of quantity governing, it may be stated that, owing to the reduced compression at small loads, the smooth running of the crank-gear is unfavorably influenced, and the partial vacuum formed in the cylinder, when the load is taken off the engine towards the end of the suction period, necessitates the use of very strong springs to load the valves in order to prevent them from re-opening and thus causing hammering and also prejudicing exact governing.

(*c*) *Combined Quantity and Quality Method of Governing.*—An example of a valve-gear of this kind is that of Mr. Reichenbach, constructed by the Maschinenbau A. G. Union, Essen, and by the Maschinenbau-Anstalt of Görlitz, in which, from the maximum power, down to a certain power, only the amount of the constant mixture is varied, while from this power, down to the power when the engine is running unloaded, proportionately more air is added; that is, the mixture is weakened, in order not to allow the compression to fall too much when running without load. In order to assure the ignition and combustion of the weaker mixture, at low powers, Reichenbach allows, with small loads, the moment of ignition to be so regulated by the governor, that the ignition, from the beginning of the weakening of the mixture, with a diminishing load, takes place earlier.

By this valve-gear, which will certainly prove effective, the air as well as the gas, each separately, then the mixture formed, and finally the ignition, should all be controlled by one or two governors.

(*d*) *Governing with a Constant Mixture and Constant Compression.*—A governor working in this manner has been patented by me and constructed by Schüchtermann & Kremer (Fig. 12). It was constructed in answer to a demand by Professor Meyer[14]

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[14] *Stahl und Eisen*, vol. xxv., pp. 132 to 144 (1905).

for a method of arranging the mixture which, at constant compression, and with increasing quantity of air, renders complete

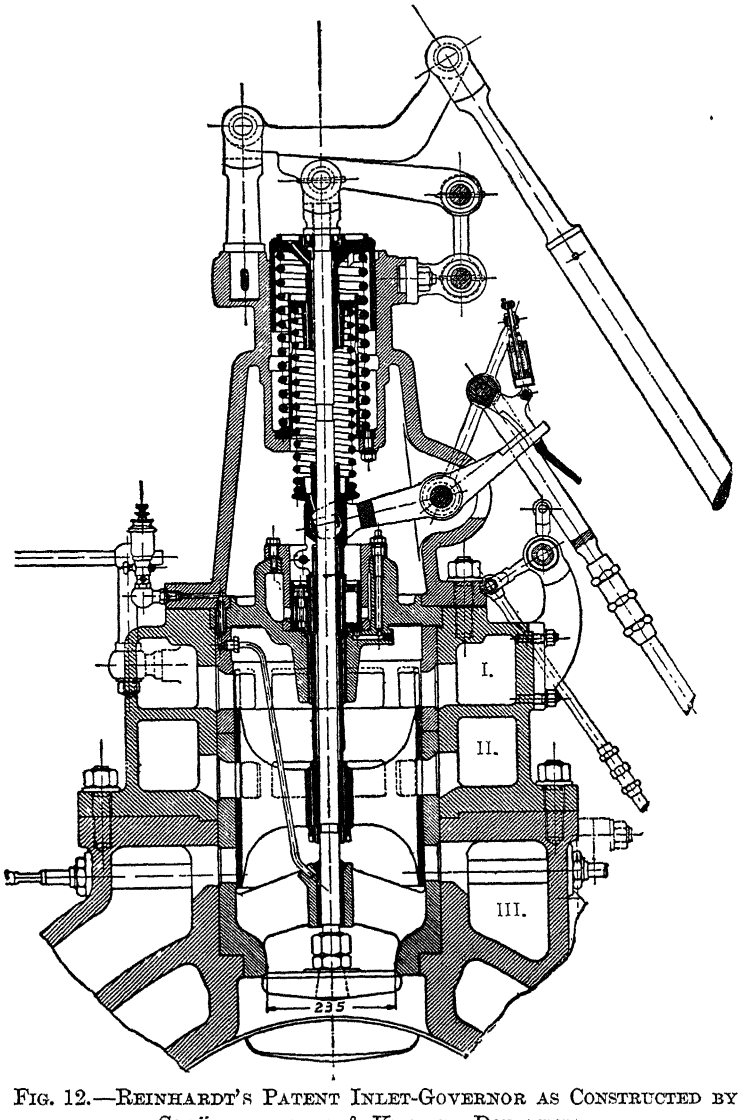


FIG. 12.—REINHARDT'S PATENT INLET-GOVERNOR AS CONSTRUCTED BY SCHÜCHTERMANN & KREMER, DORTMUND.

combustion possible, even when the engine is running without load.

The disposition of this governor consists therein, that two separate air-ports and a gas-port lead into the cylindrical space above the inlet-valve.

The inlet-valve opens at the commencement of, and closes at the end of, the suction-stroke. In the cylindrical chamber above the inlet-valve, and independently of it, a slide moves in such a manner that it first keeps the gas-port, I., and then one of the air-ports, II., shut, while it allows the admission of pure air through the air-port, III., until, at a position of the piston depending on the load at that moment, influenced by the governor, it is suddenly disconnected from its outer mechanism, and through its resulting rapid downward motion, suddenly closes the air-port, III., at the same time, however, opening the air-port, II., and the gas-port, I., so that both air and gas enter for the mixture, both from rest, and through areas which are of correct proportions. Only after the inlet-valve is closed does the slide again move upwards.

3. *Stuffing-Boxes, Cooled Pistons, and Piston-Rods.*

These important parts of large double-acting motors offer at the present time less difficulties than could ever have been expected.

There are stuffing-boxes of various constructions in use, all of which give satisfaction, and the following packings may be cited as examples: Sieger (Fig. 13), Maschinenbau-Gesellschaft Nürnberg (Fig. 14), Elsässische Maschinenbau-Gesellschaft (Fig. 15).

The construction of these packings can be clearly seen from the figures, and requires no special description.

With several packings all the rings are made of cast-iron. In a few types only those rings situated nearest to the explosion-chamber are of cast-iron, while the remaining rings are made of suitable white metal. Several packings have an extra front-packing—*e.g.*, in the Howaldt packing.

Most packings permit a movement of the packing-rings in a direction perpendicular to the axis of the cylinder only; a few others also allow a slightly inclined motion of the rod.

Great care must be taken that the cylinder-cover is well cooled, that the packing-rings are well lubricated, and that they have never to support the weight of the piston-rod. This might happen, however, if, in the course of time, the clearance between the packing-rings and the stuffing-box should

become filled with burnt residues. Therefore it is necessary, from time to time, to remove the packing for cleaning, and for this reason it is advisable to make the stuffing-box a separate and easily removable part (Fig. 13), and not continuous with the cover.

The following description of the various types of gas-engines shows that the pistons, cooled through the hollow piston-rods, differ in construction. They have proved very difficult to design; pistons broke both when they were made in one or more pieces and when they were low or high in tensile strength. With the thicknesses of walls, which are necessary to transmit

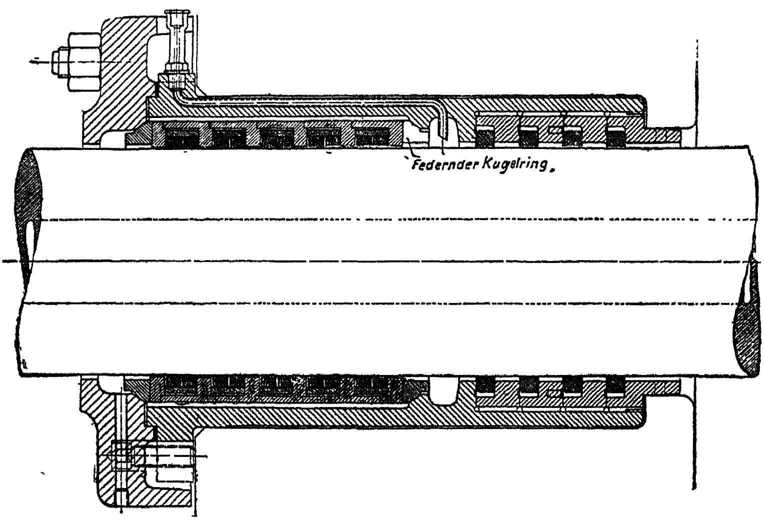


FIG. 13.—THE SIEGER PACKING.

the energy of the explosion, the initial stresses in pistons are already dangerous, wherefore it is necessary to reheat the cast-steel pistons after casting. It is, moreover, not advisable to stiffen them with ribs, as these, as is the case with cylinder-heads and cylinder-covers, often are the cause of fracture. With pistons divided into two parts, great attention must be given to the joint to prevent leakage of the cooling-water, which is under pressure of from 3 to 5 atmospheres, at the circumference of the piston, as even the smallest leakage prevents the formation of the electric spark necessary for ignition.

Finally, the fixing of the piston on the rod is a very important point.

The old-fashioned method of securing the piston to the piston-rod by a screwed end and a nut may be employed if the materials of the rod and of the nut are of very different hardnesses; otherwise, as proved by experience, a slackening of the nut is often impossible. The most practical design for this purpose is certainly that first constructed by Cockerill, in which the two halves of the piston are pressed against a flange, forged on the piston-rod, by small screws, which can easily be slackened.

The cooling of the piston-rod and of the piston is now gen-

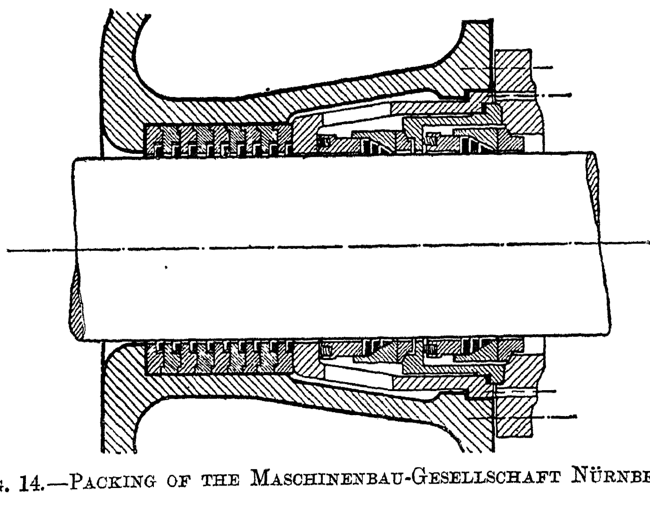


FIG. 14.—PACKING OF THE MASCHINENBAU-GESELLSCHAFT NÜRNBERG.

erally so arranged that the cooling-water enters the rod at one end and flows out at the other. A flowing-back is avoided by a pipe being fitted in the bore of the piston-rod. In tandem-engines this arrangement is either on each cylinder, or the cooling-water is allowed to pass through both rods and both pistons, one after the other. In the first case the cooling-water must be at a pressure of from 2.5 to 3 atmospheres, and in the second from 4.5 to 5 atmospheres. Concerning the manufacture of the piston-rod, that system is naturally the best by which the axis of the piston-rod, when erected and loaded with the pistons and the water they contain, is a straight line. In order to

attain this end the piston-rod, loaded in this manner, can be turned by keeping the rod fixed and allowing the tool to turn, or the rod is turned with the lathe-centers displaced in such a manner, that at the middle point of a line joining the centers of the end-sections the rod has a deviation which is equal to the deflection of the rod when loaded.

This method is not generally adopted in two-cycle engines, because the piston is too long, being equal to the stroke of the engine; it is also too heavy. The piston is allowed to rest on the

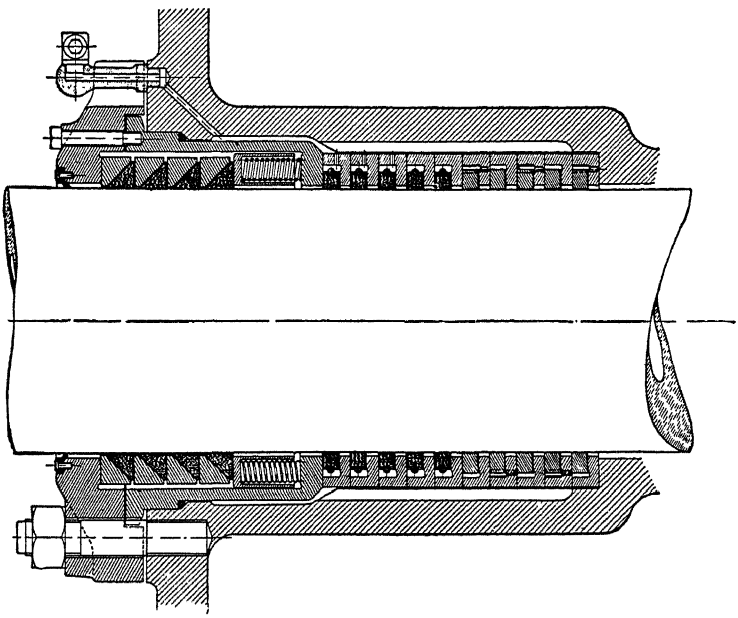


FIG. 15.—PACKING OF THE ELSÄSSISCHE MASCHINENBAU-GESELLSCHAFT, MÜLHAUSEN.

cylinder at the risk of greater wear of the latter. Although it is evident that a free-bearing piston, which does not unduly load the working-surfaces, and a piston-rod working straight and not arched, are of great value in minimizing wear of the cylinder and maintaining tightness in the stuffing-boxes, yet, on the other hand, the danger of wear of the cylinder through the weight of the piston in two-cycle engines must not be exaggerated. This wear is in any case much greater at the edges of the piston-rings, which often require replacing in large num-

bers, especially when they are made with too strong a spring, than that arising from the weight of the piston. This follows also from experience which was gained with the old four-cycle motors, in which the piston at the same time formed the cross-head, and thereby, in addition to the load due to its own weight, transferred the much greater guiding-pressure to the surface of the cylinder. In these engines wear was principally encountered at that part of the cylinder over which the piston-rings passed and the front portion of the cylinder, but the piston itself showed hardly any signs of wear. Therefore, not more piston-rings should be used than are necessary to keep the long double-cycle pistons tight; moreover, it is better to distribute them at both ends of the piston.

The lower surface of the cylinder, which should be well lubricated, would wear better if the exhaust ports were omitted, for much of the oil blows out through them.

4. *Ignition and Starting.*

A magneto-electric apparatus, driven by the engine, is generally employed for producing the electric spark to ignite the mixture at the end of the compression-stroke. These have in all cases given satisfaction. The induction-spark, by the aid of an accumulator, as employed by the Maschinenbau-Gesellschaft Nürnberg, is also satisfactory. Frequently two igniters at each end of the cylinder are fitted to insure safety and rapidity of the ignition and combustion even should one be out of order. The ignition-plugs—which are fitted in the combustion-chamber of the engine, and there carry the levers, by the separation of which the contact is broken and the spark created—were formerly cooled by a circulation of water. This has, however, been found to be unnecessary, and the plugs can now be easily removed without disturbing the water-connections.

The rapid removal ot plugs is important, because the presence of bad gas and the non-production of the spark are the principal causes, in modern engines, of a refusal to start; happily this does not often occur. If the magneto-electric apparatus is in good order, it is clearly indicated that the plug is covered with moisture, and hence no spark can be originated.

Dampness can be deposited during the night when the en-

gine is not running, also when the admission- and exhaust-valves are open. In starting it may be condensed and settle from the compressed air used, if this contains moisture. In many plants the rule is to remove the plugs each time the engines are started and thoroughly heat them.

To prevent water or moist compressed air being carried over from the air-holder, care must be taken to drain the latter, also to take the air from the highest point of the holder.

Should ignition fail at one end of the cylinder while the engine is working, this requires the driver's special attention. This failure may be occasioned by a leakage of the cooling-water from the piston, at a pressure of from 3 to 5 atmospheres, by the partial fracture of the piston, of the walls of the cylinder, or of the cover. This water, leaking out during the suction period, squirts against the plugs on the return stroke of the piston.

In such cases, when the driver is convinced that the outer ignition apparatus is in good order, the engine must be stopped and the reason of the ignition failure ascertained; also, if the load on the engine will allow of it being done, one end or one cylinder should be put out of service. If, however, they cannot be spared, at any rate the gas in the cylinder concerned must be shut off, and the compression working cut out, for instance, by wedging up the exhaust-valve. If it is supposed that the piston is cracked, even though the leakage be very slight, the cylinder should only be kept at work in case of great necessity, because the presence of water in the cylinder quickly causes considerable wear. If the precaution is taken to heat the ignition-plugs before starting, and, moreover, to make sure that the gas is suitable and burning with a steady bluish flame, the starting of gas-engines no longer offers the slightest difficulty. Further, since the general adoption of compressed air for starting large gas-engines, the time has passed when hours, and even days, were spent in vain efforts to make the engine start.

The pressure of the air employed ranges from 6 to 25 atmospheres. In most cases the valves work in the same cycle when starting as when running. The compressed air is admitted at what would usually be the commencement of the combustion-stroke and gives the engine a start. The moment of admission

of the compressed air should be determined in consideration of the fact that, in case of an ignition of the gases now drawn in, the combustion-pressure attained is higher than that of the compressed air. Further, no such admission should take place before or during combustion, as it would deteriorate the mixture, In multiple-cylinder engines, particularly two-cycle engines, which can start with a correspondingly small load, starting is often possible by admitting compressed air to one cylinder. In such cases ignition must be allowed to take place in the second cylinder, then the compressed air must be shut off in the first cylinder, and then after a few revolutions, and after the moisture originating from the compressed air has been evaporated by the heat developed by compression, the gas-valve in the first cylinder must also be opened. In starting gas-engines the ignition mechanism must be so arranged that ignition of the mixture takes place at a time which corresponds to a smaller crank-angle, distant from the dead center, than obtains at the regular speed. In the same manner the ignition must also be regulated by hand if the number of revolutions of the engine is variable, as is the case with gas-blowers.

V. The Various Types.

1. *Double-Acting Four-Cycle Engine of the Maschinenbau-Gesellschaft Nürnberg.* (Figs. 16, 17, 18, 19, 20, 21, 22, and Plate I.)

This firm and their licensees, Haniel & Lueg, Düsseldorf, and Friedrich-Wilhelmshütte, Mülheim on the Ruhr, have, if size and horse-power are taken into consideration, constructed the greatest proportion of large gas-engines in Germany, including the largest units—namely, twin tandem engines from 3,600 to 4,000 brake horse-power.

As shown by the Nürnberg single-cylinder engine (Figs. 16, 17, 18), and by the tandem engine driving a blowing-cylinder also in tandem (Plate I.), the design is very similar to that of steam-engines.

The graceful form and the careful design of the principal parts of the Nürnberg engines are clear from the figures, and require no further comment.

The frame of the engine is open at the top to render the

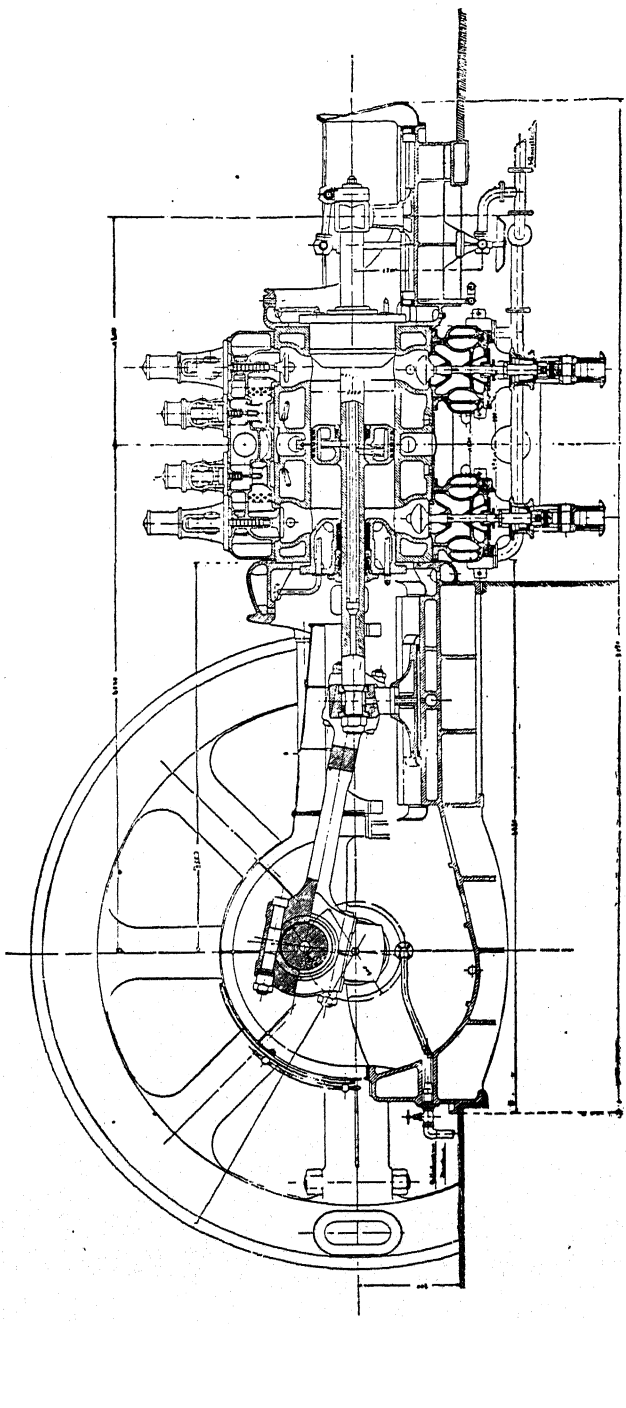


FIG. 16.—SINGLE-CYLINDER ENGINE OF THE MASCHINENBAU-GESELLSCHAFT NÜRNBERG.

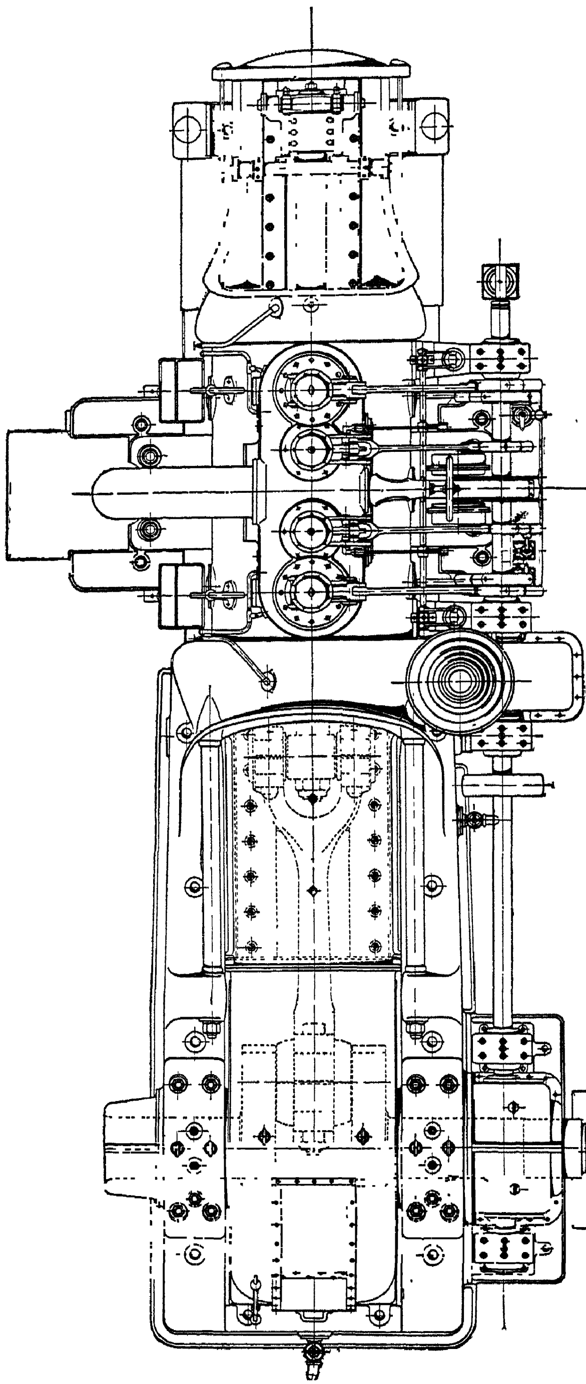


FIG. 17.—GROUND-PLAN OF THE SINGLE-CYLINDER ENGINE OF THE MASCHINENBAU-GESELLSCHAFT NÜRNBERG.

crosshead, stuffing-box, and cylinder-cover accessible; but the opening is entirely covered with a plate when the engine is running. The open frame terminates in a strong circular flange to form a concentric point with the cylinder; the flange

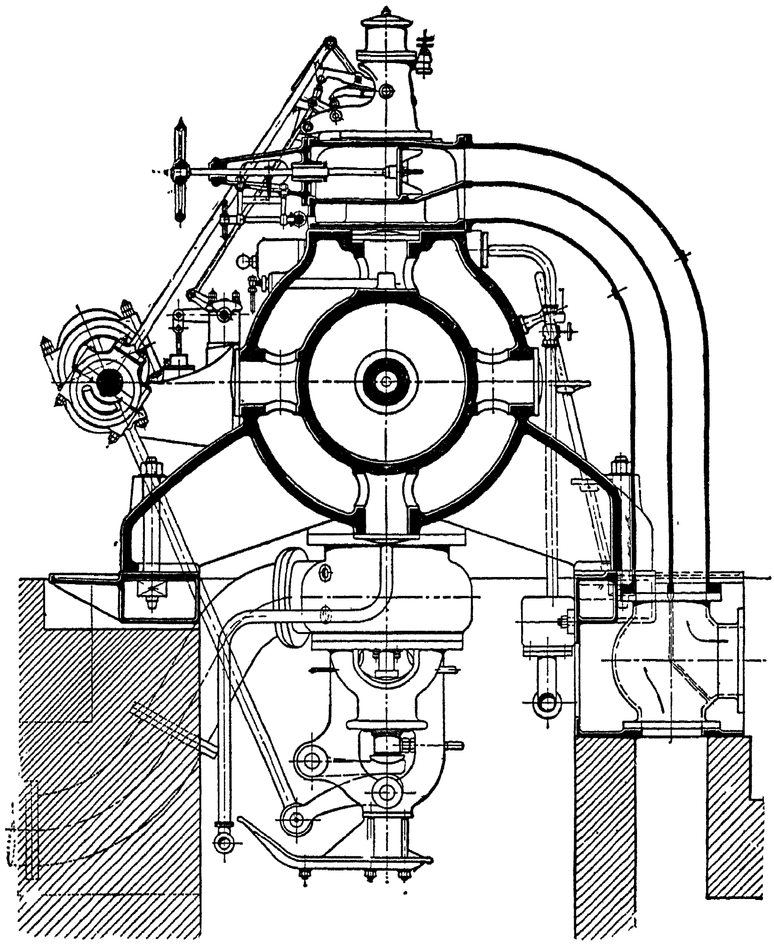


FIG. 18.—CROSS-SECTION OF CYLINDER OF THE MASCHINENBAU-GESELLSCHAFT NÜRNBERG.

is strengthened by tension-rods extending to the part which forms the crank shaft-bearing. The distance-pieces, which are also strengthened by tension-rods, unite the separate cylinders in a similar concentric manner. Owing to this method of con-

struction the erection of the engine is easy and exact. The lower portions of the distance-pieces (Plate I.) form the cross-head guides for supporting the piston-rods; and the upper portion is provided with an opening through which the cylinder-cover and piston can be removed.

The cylinders are entirely symmetrical, with wide flanges forming large cooling-spaces. A number of examination-doors are provided for clearing out, when necessary, any deposit. At each cylinder-end there is an inlet-valve above and an exhaust-valve below; and these valves are situated well outward from the cylinder-walls, so that the highest temperatures occurring have to extend their unfavorable influence along the axes of the valves to the outer valve-boxes.

The gearing of all the valves is operated by an eccentric and roller-levers. The gas-valves are situated in the longitudinal axis of the cylinder close to the inlet-valves, and their motion is controlled by the governor.

For the method of governing and the formation of the mixture and of the exhaust-valve chambers, the author would refer to what has already been mentioned (p. 698).

The pistons are cast hollow but in one piece, and are pressed on to the conical part of the rod by nuts recessed into the pistons. By this means the front and back cylinder-covers can be made symmetrical.

The lubrication of all the working-parts is arranged in an effective manner from a central position.

Fig. 19 shows a Nürnberg tandem engine with the front cylinder-covers removed, and prepared ready for cleaning the front valves.

With the front cover removed the front exhaust-valve is accessible from the crosshead guide and under the piston-rod. Attention is, however, drawn to the fact that, after having screwed off the cylinder-cover and taken off the heavy cover in the manner described, this valve is accessible, although not altogether easily. Much stress should not be laid on this accessibility with cylinders of less than 500 horse-power.

This applies to all systems in which the exhaust-valves are placed below the piston-rod.

The cleaner the gas and the purer the water for cooling the

valves, the less important is it that the valves should be easily accessible.

The construction of the blowing-cylinder of the Maschinenbau-Gesellschaft Nürnberg can be seen from Plate I.

The inlet-valves are controlled by an eccentric on the crank-shaft and a link-motion with long rods, in such a manner that, with an equal power of the engine, should the air-pressure become higher than usual, a smaller quantity of air is drawn in, also giving relief when starting. The pressure-valve gear consists of valves preceded by slide-valves, so controlled that they open much earlier than the pressure-valves, but at the dead center they close and thus give the pressure-valves (which are loaded with light springs) time to close. Seeing that while the volumetric efficiency, both with isothermal and adiabatic compression, is without influence on the work to be done per unit of the quantity of air compressed, the heavy and copious lubrication required by pressure slide-valves may only favor the use of valves with a long lift and with light spring pressure.

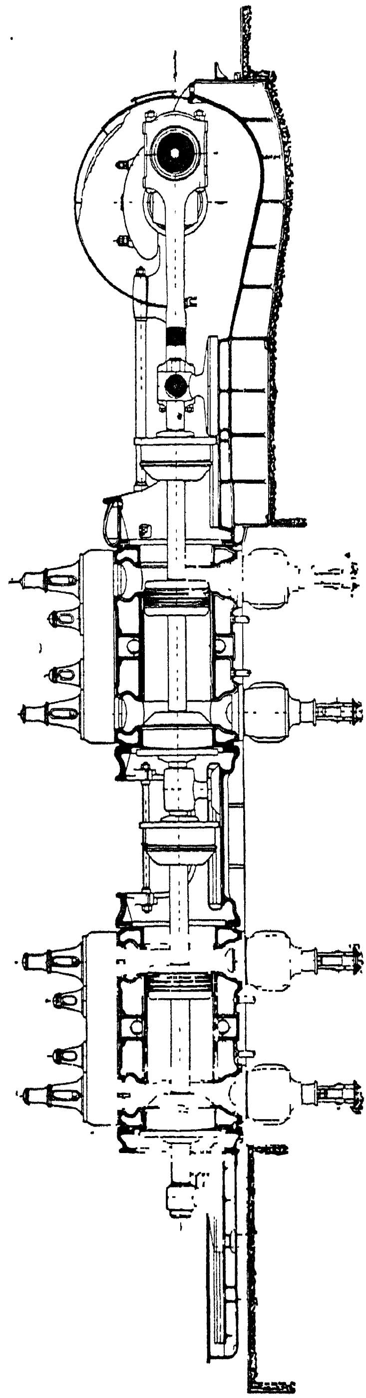


FIG. 19.—CLEANING THE FORWARD VALVES OF THE NÜRNBERG GAS-ENGINE.

2. *Double-Acting, Four-Cycle Engine of the Gasmotorenfabrik Deutz.* Figs. 23, 24, 25, 26, and Plate II.

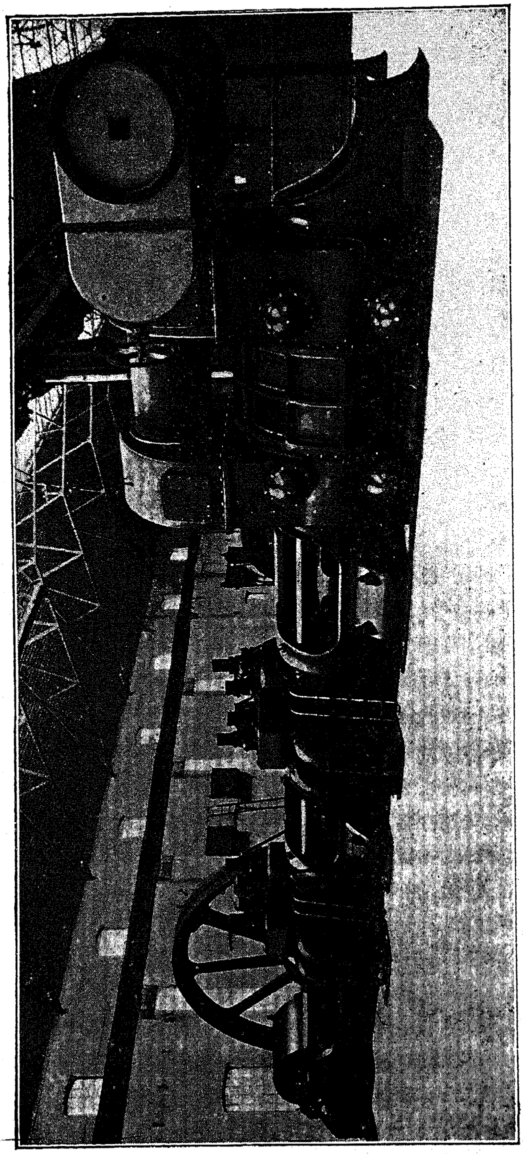


FIG. 20.—BLAST-FURNACE BLOWING-ENGINE OF 950 EFFECTIVE H.P., 80 REVOLUTIONS, BUILT FOR THE ROMBACHER HÜTTENWERKE BY HANIEL & LUEG, DÜSSELDORF.

The Gasmotorenfabrik Deutz engines have lately been described by Professor Meyer.[15]

The details of the cylinder of a 250 h.p. single-cylinder en-

[5] *Stahl und Eisen*, vol. xxv., pp. 67 to 72, 132 to 144 (1905).

gine have already been described. This was the first engine of this type constructed by the Gasmotorenfabrik Deutz. Having regard to what has already been stated, the peculiarities of the

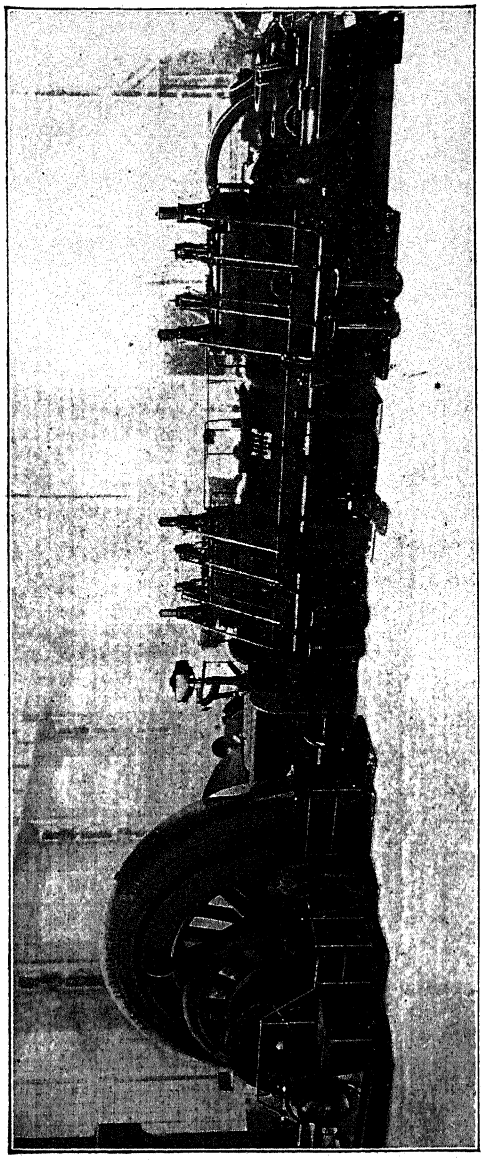


Fig. 21.—Gas-Engine of 1,500 to 1,800 Effective H.P., 94 Revolutions, Built for the Schalker Gruben-Hütten Verein, Gelsenkirchen, by the Maschinenbau-Gesellschaft Nürnberg.

construction require no further explanation. It may be remarked that in these smaller engines the guides are of the circular inclosed type (Fig. 23).

The valve-gear of this engine is actuated by cams, and the quantity governing (with a constant mixture) is attained by throttling, by the governor working with a movable fulcrum

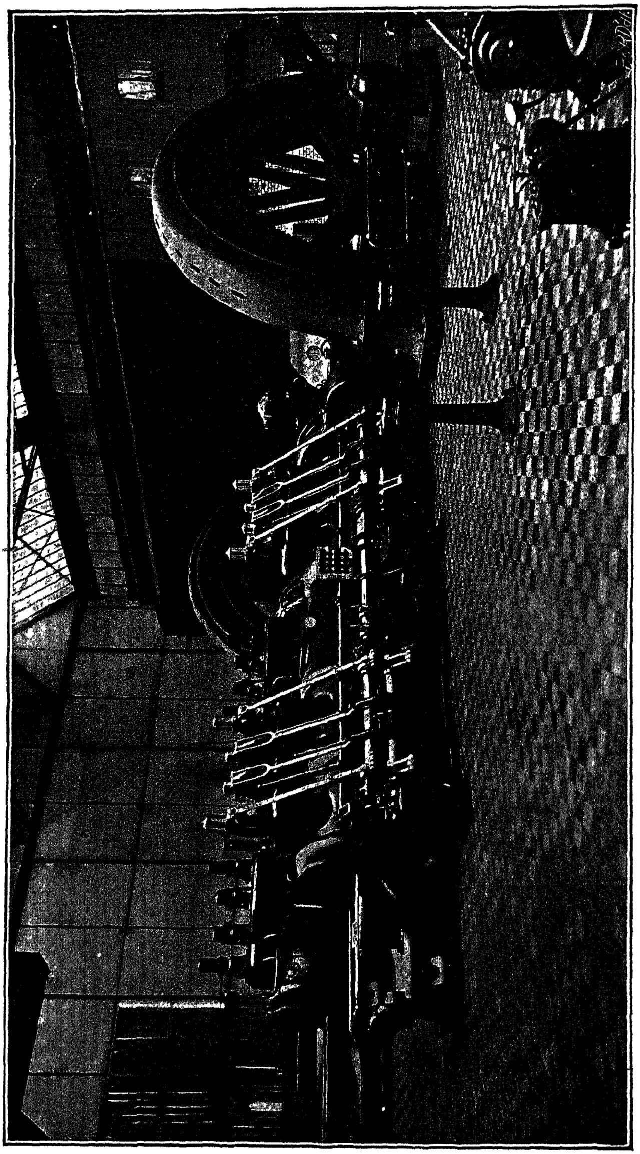


Fig. 22.—Gas-Dynamo of 600 Effective H.P., 125 Revolutions, Built for the Moselhütte Act.-Ges., Maizières, by Haniel & Lueg, Düsseldorf.

on the lever of the inlet-valve, by which means the lift of the inlet-valve is increased or reduced. This arrangement of the valve-gear is simple, as no special gas-valve has to be controlled.

It would not, however, be suitable for engines with two or more cylinders, say with four inlet-valves, for the movable fulcrum of at least one valve-lever would be fixed by the action of a strong valve-spring, and thereby the governor would encounter excessive resistance. For this reason the Gasmotorenfabrik Deutz, in their latest engines, have placed, at the side of the

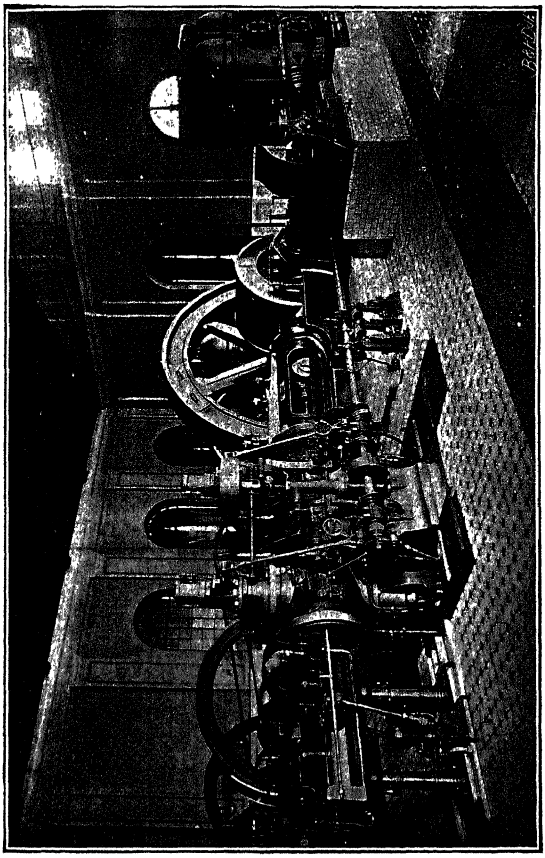


FIG. 23.—250-H.P. GAS-ENGINE OF THE GASMOTORENFABRIK DEUTZ.

principal valve, a special mixing-valve for the admission of gas and air, so that when the governor operates, the fulcrum of the lever of the main valve is fixed while that of the mixing-valve is moved (Fig. 24), thereby considerably reducing the resistance to the governor, and, moreover, the mixing-valve is rendered accessible for cleaning.

Seeing that the mixing-valve is controlled by the mechanism

of the main valve, the gear is simple. From Fig. 24 it will be further seen that the Gasmotorenfabrik Deutz has introduced a

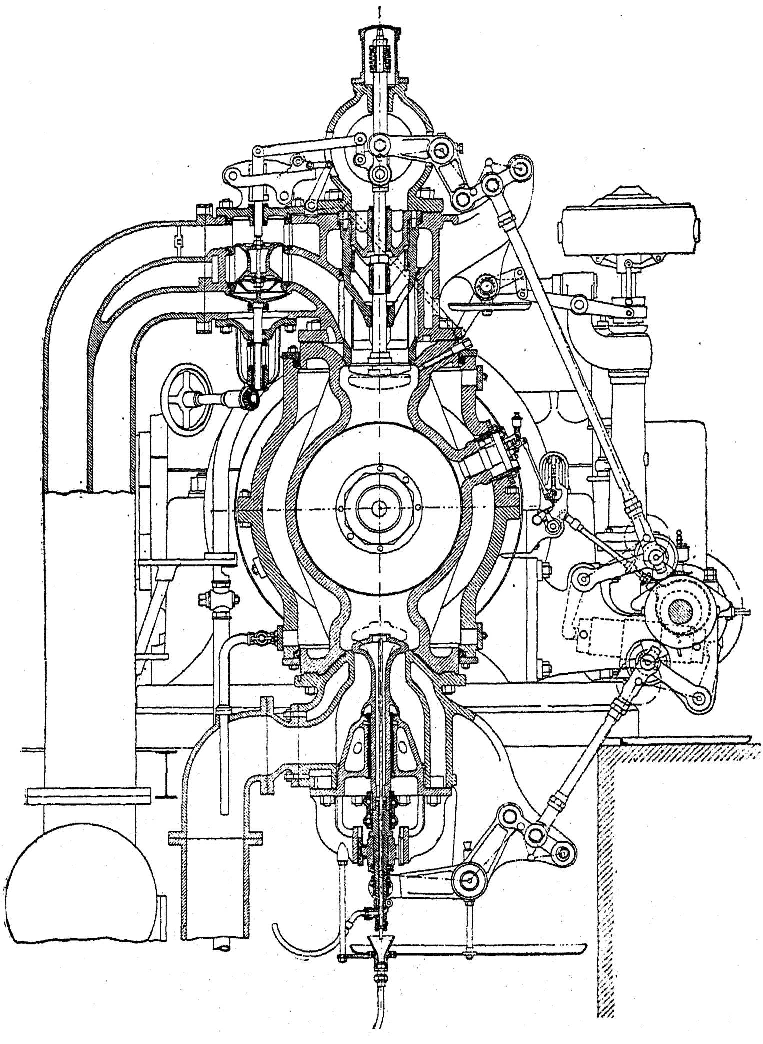


FIG. 24.—NEWER INLET- AND OUTLET-VALVE REGULATION OF THE GASMOTORENFABRIK DEUTZ.

patented arrangement of a bell-crank lever in front of the valve-levers, which, on the one hand, replaces the action of rolling-levers in lifting and closing the valves, and, on the other hand,

prevents undesirable opening of the valves, which may occur when the engine is lightly loaded or running without load.

When the valve is closed the bell-crank lever is in its limiting position, so that the forces transmitted by the valves can cause no bending of the bell-crank lever, and therefore no motion. As shown in Fig. 24, the Gasmotorenfabrik Deutz now makes a complete separation of the inner and outer cylinder walls at the intersection with the valve-chambers, whereby the stresses due to heat are reduced.

The arrangement of the governor-gearing of the Deutz engine here described gives only a downward resistance for the fulcrum of the lever of the mixing-valve. Should the valve-spindle or the spindle-guides become dirty and stick, the weight of the mixing-valve and the pressure of the spring are no longer sufficient to overcome the resistance, and the action of the governor will be unreliable.

In the construction of their 2,000-h.p. tandem engine (Fig. 25, Plate II.) this possibility is, however, removed, because, in this engine, the lever of the mixing-valve is made in the form of a closed link, which on both sides incloses the movable fulcrum. The movable fulcrum of the mixing-valve lever is thereby the end-point of a lever capable of being turned on a fixed center by the governor, the length of which represents the radius of the curve of the link. In this case the strong spring of the inlet-valve, together with the weight of the mixing-valve and its light springs, aids the closing of the valve.

The Gasmotorenfabrik Deutz construct the frames of this type of engine (Plate II.) open at the top. Each cylinder consists of a cast-iron liner, to which the cast-steel cylinder-heads are bolted by means of flanges. The outer jacket is then completed by a circular casting in two pieces.

Although this construction, in my opinion, is not safer than that in which the cylinder-heads and the liner were cast together, it has the advantage that the simple cylinder-liners can be easily made of hard cast-iron.

The valve gearing is operated by round-back cams with roller-levers (Fig. 25); at each cylinder-end there is only one cam for the simultaneous operation of the inlet- and outlet-valves.

The last-mentioned is double-seated in a manner similar to

the valves of steam-engines; but as, however, exhaust only takes place at the top seating, the deposit of dirt and cinders on the bottom seating must be considered.

The roller-levers of the inlet- and outlet-valve gearing can be put out of gear by hanging the roller-path to the levers,

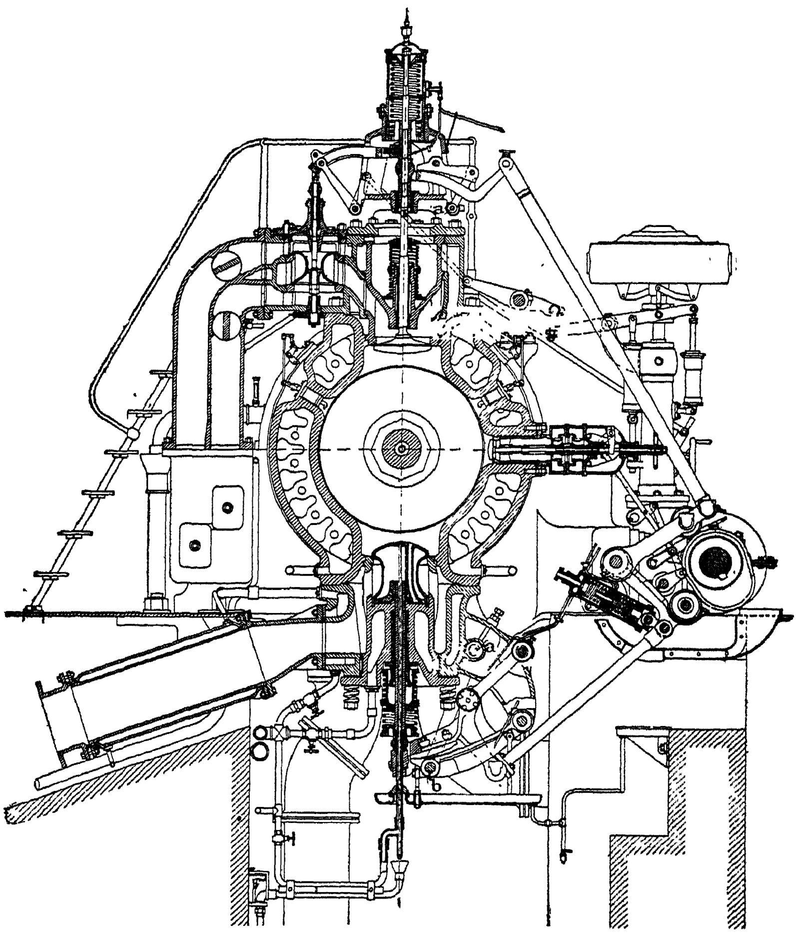


FIG. 25.—REGULATION OF INLET AND EXHAUST OF 2,000-H.P. GAS-ENGINE OF THE GASMOTORENFABRIK DEUTZ.

which can be interspersed round the eccentric pins a, a (Fig. 25). With the exhaust-valve gearing the whole system of rods can be disconnected from the outlet-valve spindle by removing a bolt, b, so that the outlet-valve with its seating can be taken down.

This engine, as represented in Fig. 26, is at work at the Hoerder Verein iron-works, and is remarkable owing to its ele-

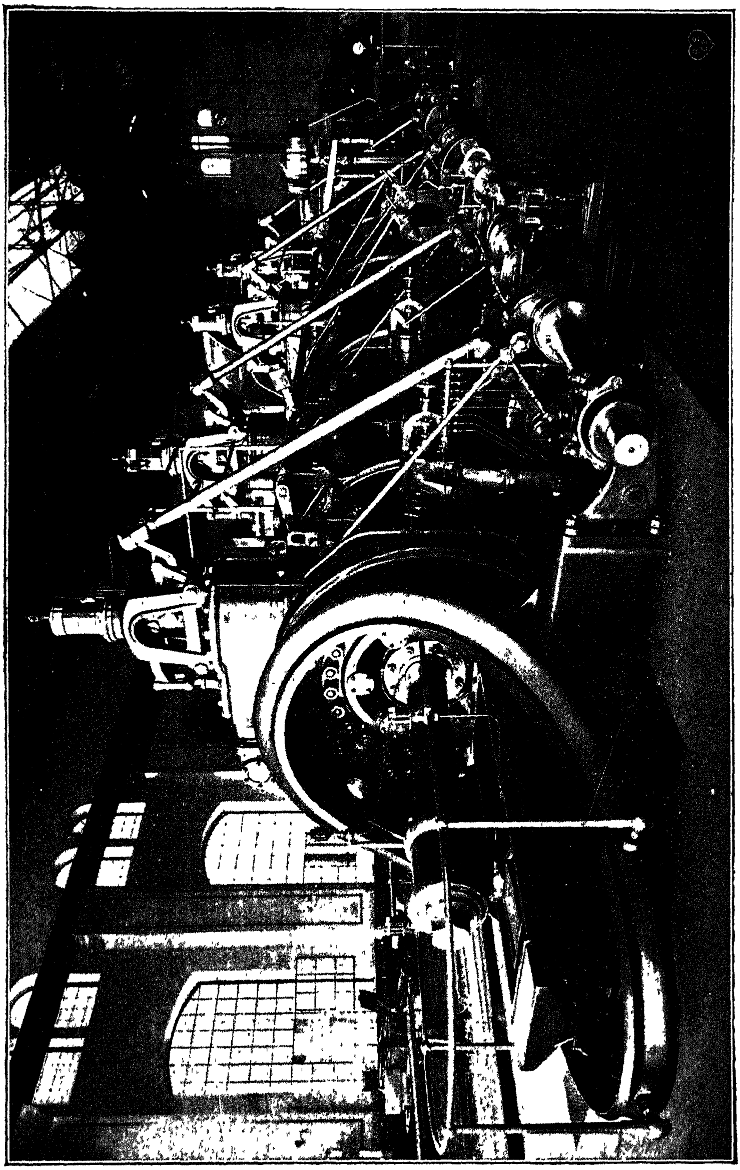


FIG. 26.—2,000-H.P. GAS-DYNAMO, BUILT FOR THE HÖRDER BERGWERKS- UND HÜTTEN-VEREIN BY THE GAS-MOTORENFABRIK DEUTZ.

gant design, strong construction, great simplicity, and faultless working.

3. *Double-Acting Four-Cycle Engine by Ehrhardt & Sehmer, Schleifmühle.* (Figs. 27, 28, 29, 30, and Plate III.)

Ehrhardt & Sehmer are licensees of the Gasmotorenfabrik Deutz. Their gas-engines are constructed, as those of the

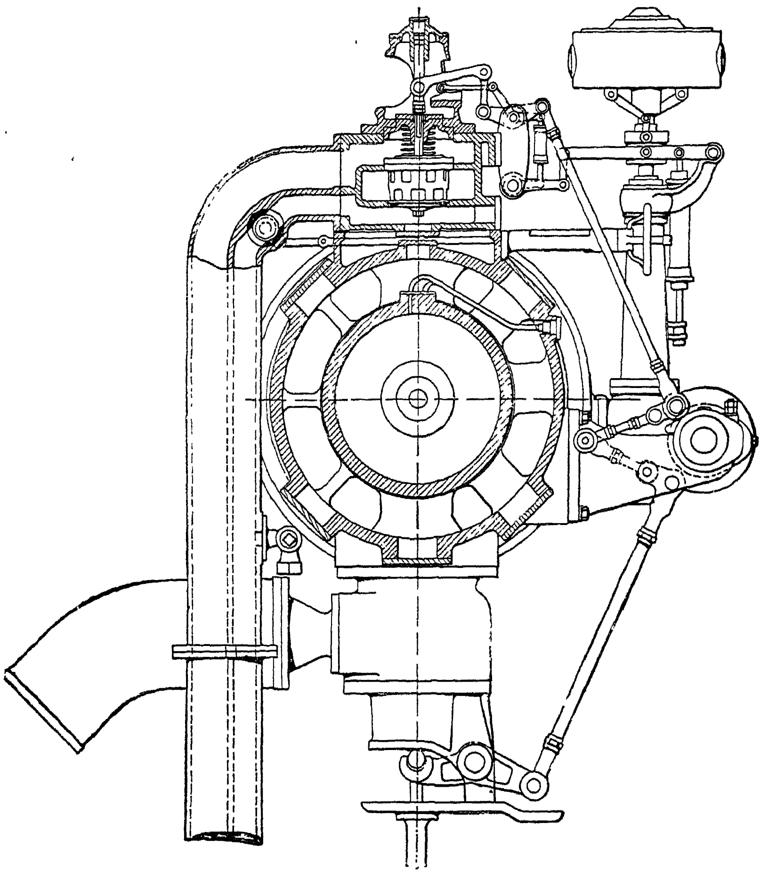


FIG. 27.—OPERATION OF MIXING-VALVE OF EHRHARDT & SEHMER, SCHLEIFMÜHLE.

latter, with quantity governing, but in other respects they have not retained the outer arrangement of the Deutz engine; for instance, with respect to the operation of the mixing-valve. The inlet-valves and mixing-valves are in a similar erection in the longitudinal axis of the cylinder, as is the case with the Nürnberg engine, with this exception, that in the Ehrhardt &

Sehmer engine each valve, including the mixing-valves, is operated by a round-back cam (Figs. 27, 28).

The mixing-valves are, according to information received from the firm, arranged in such a manner that, for working

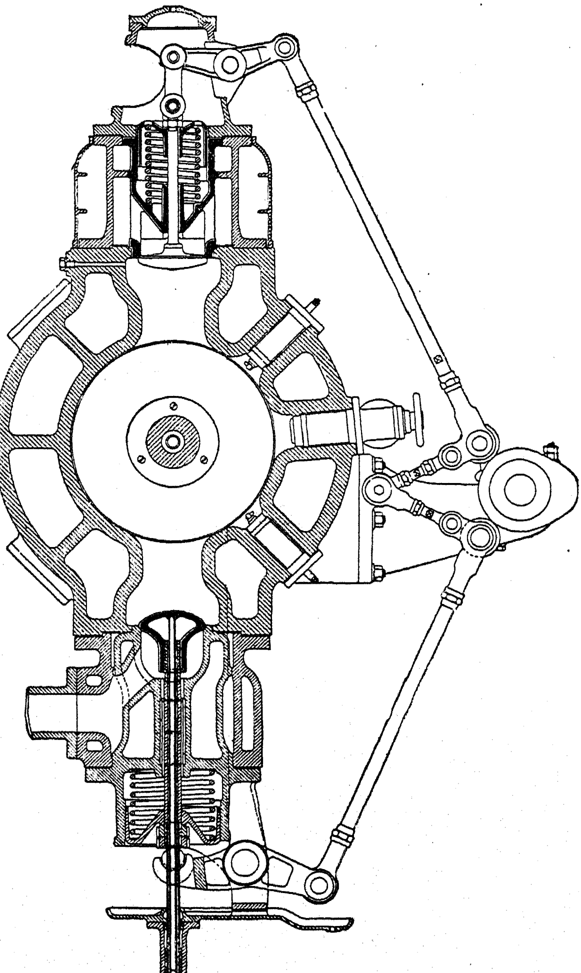


FIG. 28.—OPERATION OF INLET- AND OUTLET-VALVES OF EHRHARDT & SEHMER, SCHLEIFMÜHLE.

with various gases, the area of the valve-passage can be varied.

The cylinders have large water-cooling spaces. They and the water-jacket are cast in one piece, and are, as in the case of the Nürnberg engine, supported by the frames, by the distance-piece, between the cylinders, and by the back guide. The

outlet-valves can be removed, with their casings, without disconnecting the piping, the space under the cylinders being en-

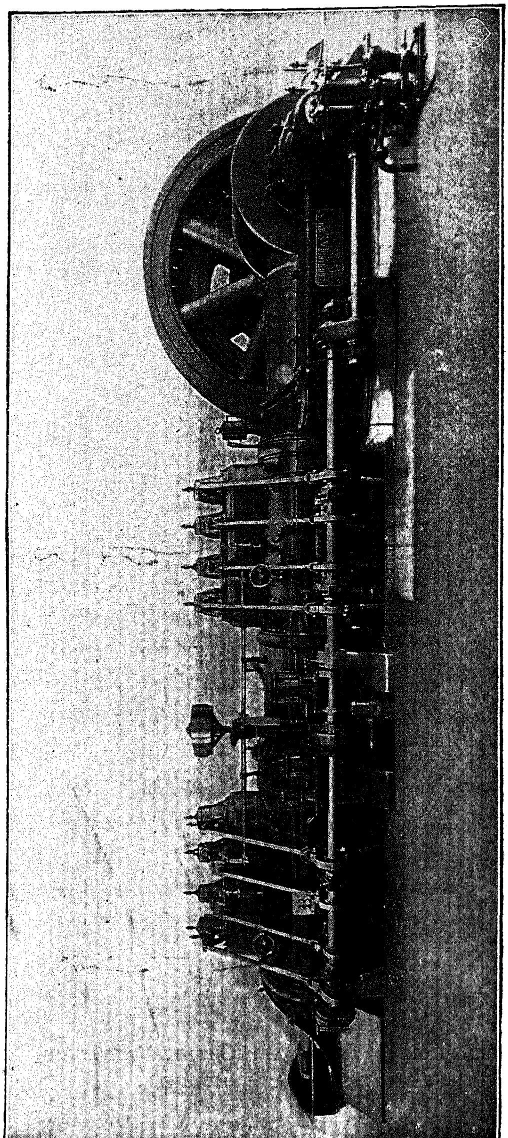


FIG. 29.—700-H.P. TANDEM GAS-ENGINE, BUILT FOR THE KÖNIGLICHE BERGINSPEKTION HEINITZ BY EHRHARDT & SEHMER, SCHLEIFMÜHLE.

tirely free. It will be seen from the figures that all the details of this engine are well made and of strong proportions.

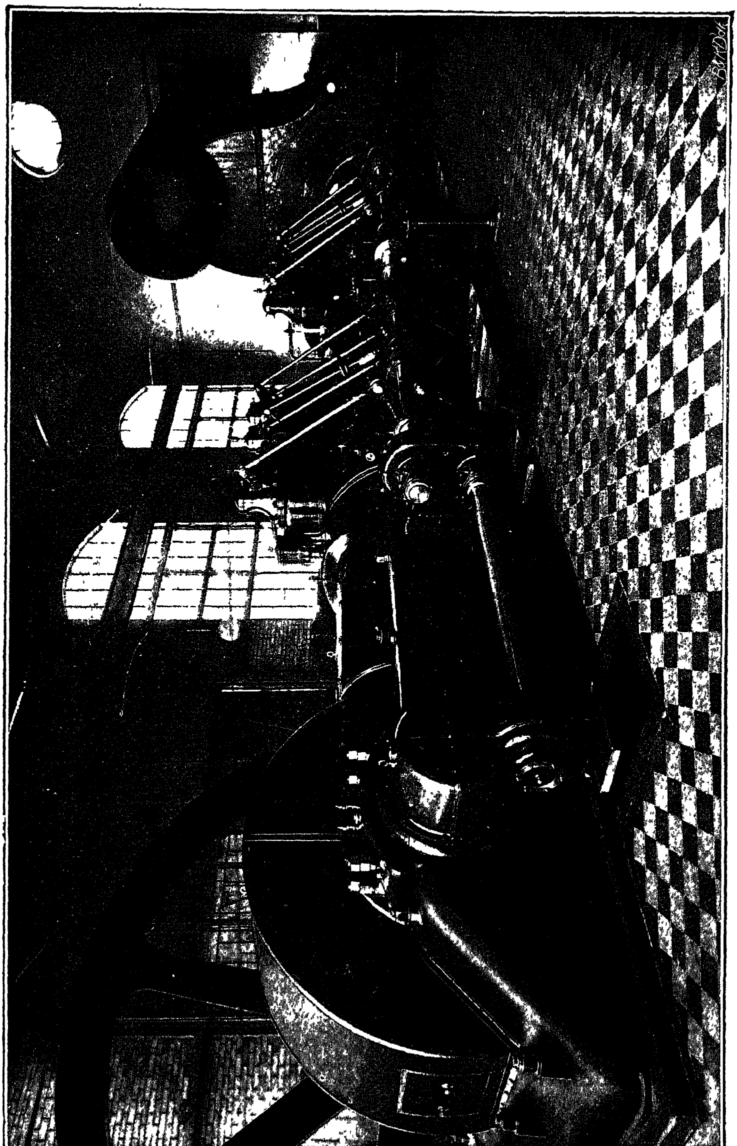


FIG. 30.—GAS-DRIVEN BLOWING-ENGINE, BUILT FOR GEBR. STUMM, G. M. B. H., NEUNKIRCHEN, BY EHRHARDT & SEHMER, SCHLEIFMÜHLE.

4. *Double-Acting Four-Cycle Engine by the Märkische Maschinenbau-Anstalt, Wetter-Ruhr.* (Plate IV.)

The Märkische Maschinenbau-Anstalt is the licensee of the Belgian firm, Cockerill, and adheres to their design.

The massive frame supports the guides between two high

cheeks, which in tandem engines are continued to the back cylinder, and the stresses are thus transferred to the crank-shaft bearing.

Less objection can be made to this manner of construction than to the retention of the class of cylinder-heads that are so liable to fracture—even though they are closed with a cylinder-cover—and to the construction of such cylinder-heads, each cast in one piece with the cheeks of the frame.

Cockerill themselves discarded this construction in the engines they exhibited at the International Exhibition of Liège, while they no longer construct the cylinder-heads, nor cast the cylinder together with the side-cheeks of the frame.

In Plate IV. it may be further observed that the outer casing is divided in the middle and closed by a ring, which is slipped over and made tight by short stuffing-boxes. The valves are operated by cams and roller-levers.

Up to the present time the Märkische Maschinenbau-Anstalt have built their engines with quality governing; they propose in the future to employ the quantity method of governing.

The arrangement for admitting the water for cooling the pistons is very carefully designed. Water is admitted by means of a combination of a turned tube with a telescope sliding-pipe. The water is conducted away from the piston-rod by a pipe, screwed into the piston-rod, which travels backwards and forwards in a slot in the cover of a trough. To remove the exhaust-valve the whole valve-casing must be disconnected from the pipe.

5. *Double-Acting Four-Cycle Engine by the Elsässische Maschinenbau-Gesellschaft, Mülhausen.* (Figs. 31, 32, 32A, 33, and Plate V.)

This firm originally, as the licensees of the company, also constructed engines of the Cockerill type; latterly, however, they have entirely modified their designs. This may be seen by comparing the engine of the Märkische Maschinenbau-Anstalt (Plate IV.) with the tandem blowing-engine of the Elsässische Maschinenbau-Gesellschaft (Plate V. and Fig. 31).

Two such engines, each of 1,500 h.p., have been working for a considerable time at the Differdingen iron-works, and are remarkable for their steady running, and, as may be seen from Plate V. and Fig. 32, for their well-designed forms and well-

thought-out design of all details. The frames consist of two girders, symmetrical with the longitudinal axis of the engines, joined together by cross-pieces, which join the massive flanged

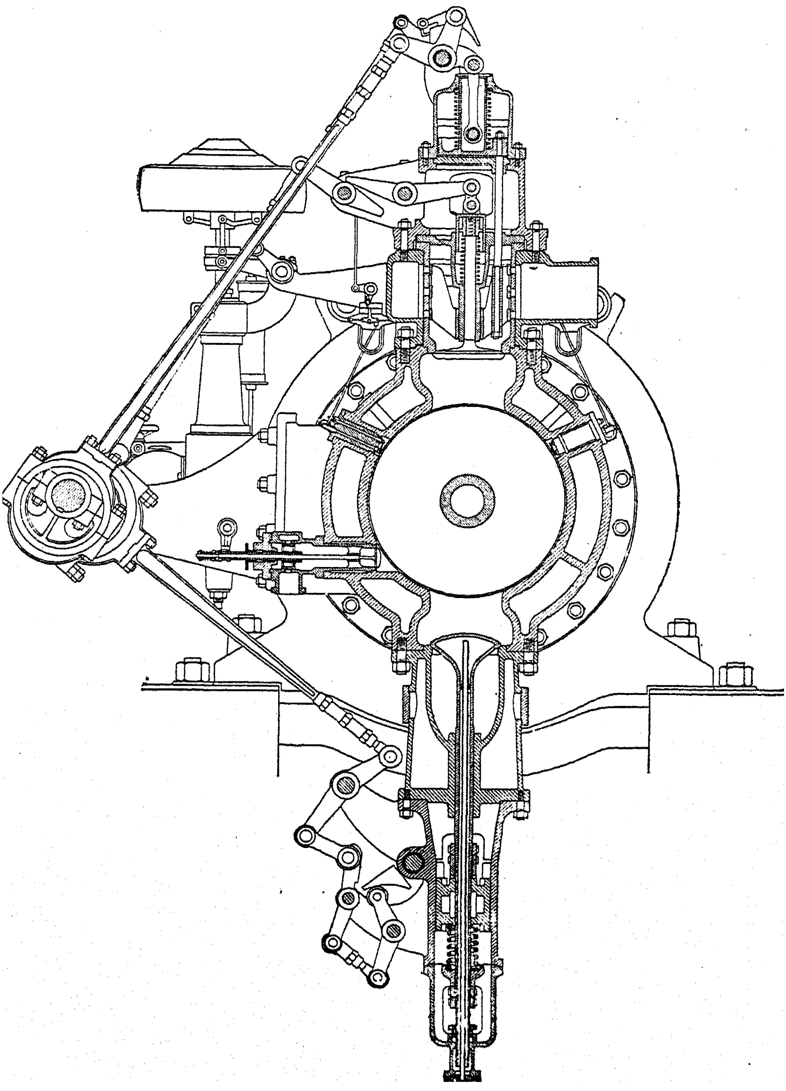


FIG. 31.—REGULATION OF GAS-CYLINDER OF THE ELSÄSSISCHE MASCHINENBAU-GESELLSCHAFT, MÜLHAUSEN.

ends of the frames to the crank-shaft bearings, whereby the stresses caused by explosion are transmitted to the crank-shaft bearing, without any bending moment, in the vertical plane

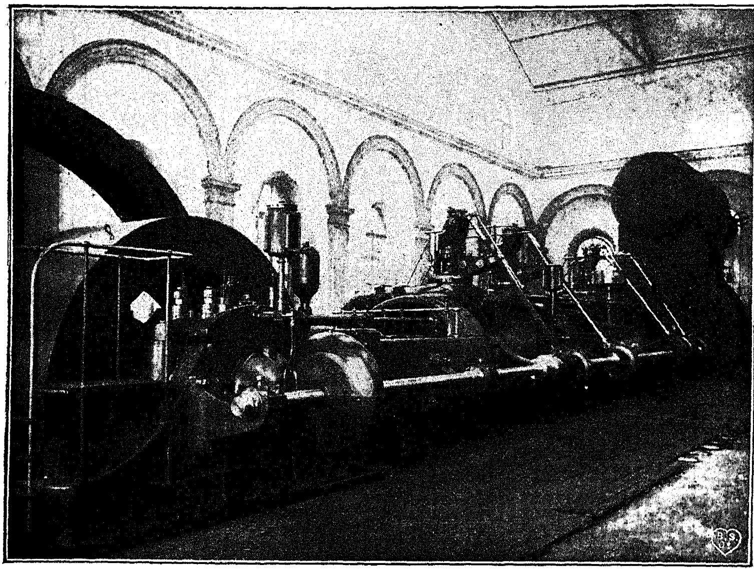


Fig. 32.

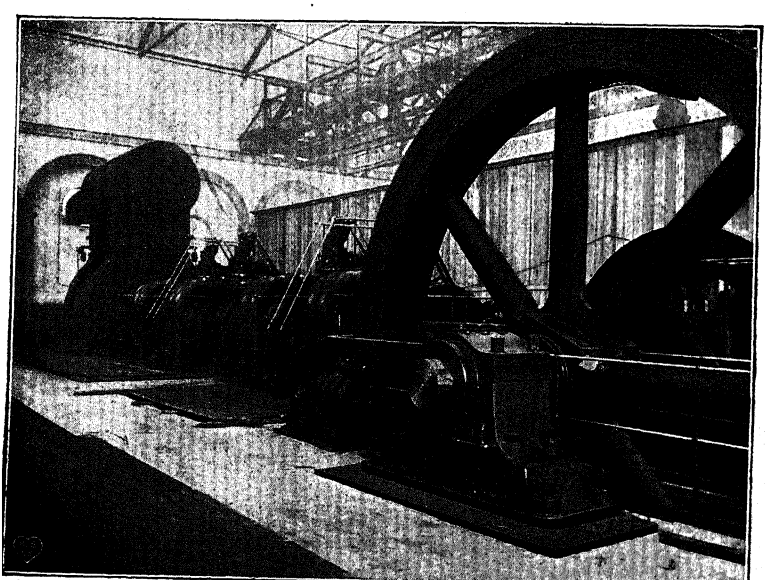


Fig. 32A.

FIGS. 32 AND 32A.—1,500 H.P. BLOWING-ENGINE, BUILT FOR THE HÜTTENWERK, DIFFERDINGEN, BY THE ELSÄSSISCHE MASCHINENBAU-GESELLSCHAFT, MÜLHAUSEN.

symmetrical to the side girders. Between these two girders, the water-cooled guides are situated.

Both the gas-cylinders and the blowing-cylinder are joined together by concentric distance-pieces, open at the top and strengthened with tie-rods; and foundation support is given only at the distance-pieces and the blowing-cylinder.

The gas-cylinders resemble those of the Gasmotorenfabrik Deutz (Fig. 7) in the manner in which they are cast. The

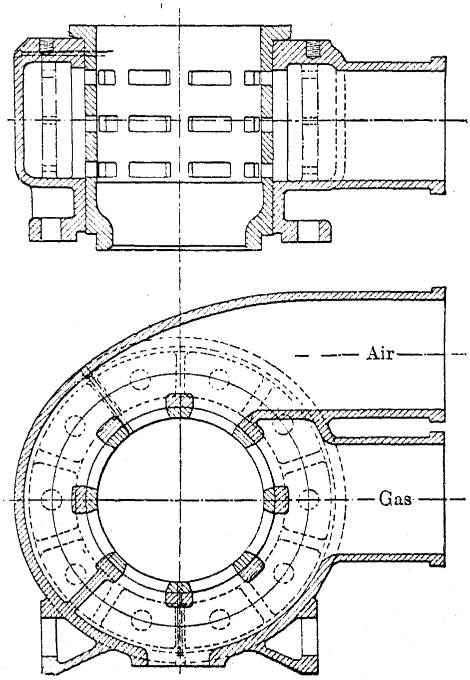


FIG. 33.—INLET SLIDE-VALVE OF THE ELSÄSSISCHE MASCHINENBAU-GESELLSCHAFT, MÜLHAUSEN.

liner and the jacket are cast together; the latter is open in the middle for about one-third of its length, and closed by a ring cast in two parts. The pistons of the gas-engines are of cast-iron, in one piece, and fixed to the rods by means of nuts. The circulation of the cooling-water takes place from a telescope-pipe, with inlet in the front piston-rod and outlet over a trough in the back distance-piece.

In this manner the pistons are water-cooled in a very simple manner, but a higher pressure is necessary than when each piston is cooled separately.

The gas-pistons can be very easily removed by detaching the crosshead, the couplings and the cover, while the front piston can be taken out at the front, and the back piston at the rear in the distance-piece between the gas- and blowing-cylinders; the piston-rod of the back gas-cylinder is pushed into the hollow piston-rod of the blowing-cylinder.

The governor regulates quantity, governing in such a manner that it causes a mixing-slide, opening with the inlet-valve, to close suddenly sooner or later (Fig. 31). So long as the mixing-slide remains open, it allows the inlet of air at one half of the chamber (Fig. 33), separated by a vertical partition, and gas at the other half. The mixture is formed by the motion of the air and gas together when passing the inlet-valve, and is quite effective.

The gearing of the inlet- and outlet-valves is controlled by an eccentric and roller-levers, and the outlet-valve is opened and closed with a restricted motion, and after closing it is kept closed by the action of only a short strong spring (Fig. 31).

As may be seen from Plate V., in order to dismount the outlet-valve the valve-casing must be unbolted from the cylinder and the pipe-connections.

The blowing-cylinder is water-cooled, and provided with suction- and pressure-valves on the Hörbiger & Rogler system, and the piston is rendered tight by two piston-rings, made in halves and lined with white metal, and pressed against the walls of the cylinder by springs. To render an increase of the air-pressure possible (in this case from 0.5 to 1 atmosphere), the covers of the blowing-cylinders are provided with chambers, which can be put into communication with the cylinder by means of valves operated by hand. Thereby the clearance-spaces are increased, and, for an equal load of the gas-engine, a correspondingly higher pressure with a smaller quantity of air per revolution is attained.

For this increase of air-pressure three steps from 0.5 to 1 atmosphere are provided for by three chambers in the covers. A fourth chamber is arranged with a special valve as a circulating-space for starting the engine with a relieved load.

6. *Double-Acting Four-Cycle Engine by Fried. Krupp Aktien Gesellschaft, Essen a. d. Ruhr.* (Figs. 34 and 35.)

This motor, which this firm have several times constructed for their own use, is worthy of notice, owing to the arrangement of the valves and the construction of the cylinder.

Both the inlet- and the outlet-valves are situated above the cylinder, in its longitudinal axis, together in the same cast-steel valve-chamber (Fig. 34). As compared with the outlet-valve

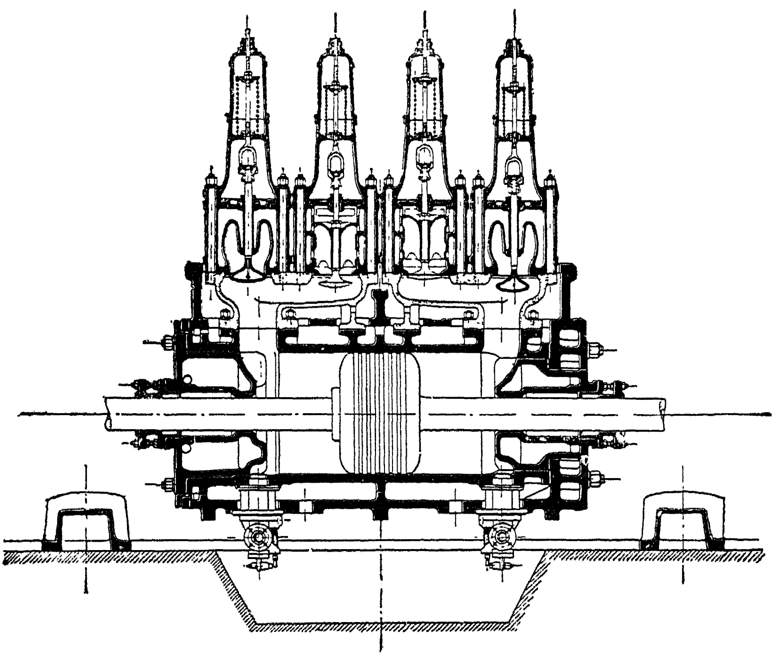


FIG. 34.—CYLINDER OF THE GAS-ENGINE OF FRIED. KRUPP AKTIEN GESELLSCHAFT, ESSEN A. D. RUHR.

situated under the cylinder, the advantage of the easy accessibility by direct lifting by a crane or traveler, of the possibility of easily examining the whole of the valve-motion, as well as of the unbroken foundation bed-plate, is very obvious.

The cylinder-liner is provided with short shoulders to receive the valve-chambers, inserted into the jacket, which is open at the top, and at the front end is bolted to the same; while at the other end the liner and the jacket are made tight by a short stuffing-box, in such a manner that the liner is free to expand independently.

In this construction of cylinder the stresses caused by heat, that usually occur when the liner and the jacket are cast together, are avoided.

The cylinder is fixed to the frame by means of massive attachments cast on to the jacket (Fig. 35). The quantity method of governing is adopted.

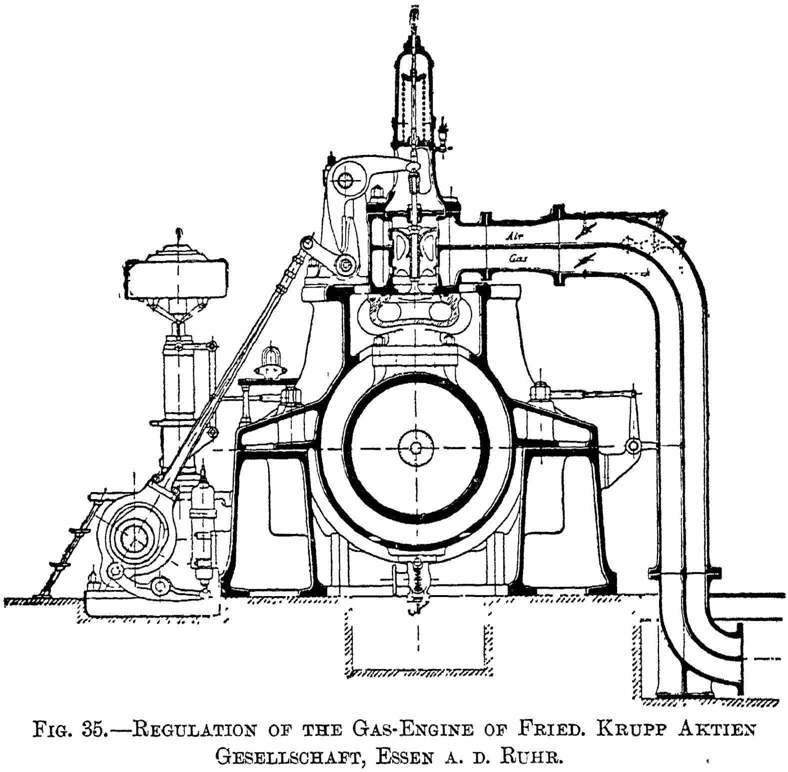


FIG. 35.—REGULATION OF THE GAS-ENGINE OF FRIED. KRUPP AKTIEN GESELLSCHAFT, ESSEN A. D. RUHR.

7. *Double-Acting Four-Cycle Engine by the Gutehoffnungshütte, Oberhausen.* (Fig. 36, and Plate VI.)

The Gutehoffnungshütte, in addition to two-cycle engines on the Körting system, also build four-cycle engines.

Their type of engine is represented in Plate VI. and Fig. 36.

The arrangement of the frame of the cylinder, cover, and distance-pieces has already in several instances been described.

One of the inlet-valves above the cylinder and one of the outlet-valves situated under the cylinder are operated by the same cam with roller-levers.

The outlet-valve can be removed without disconnecting the piping.

The mixing-valve for quantity governing (Fig. 36) is situated at the side of the inlet-valve, and is actioned by a release mech-

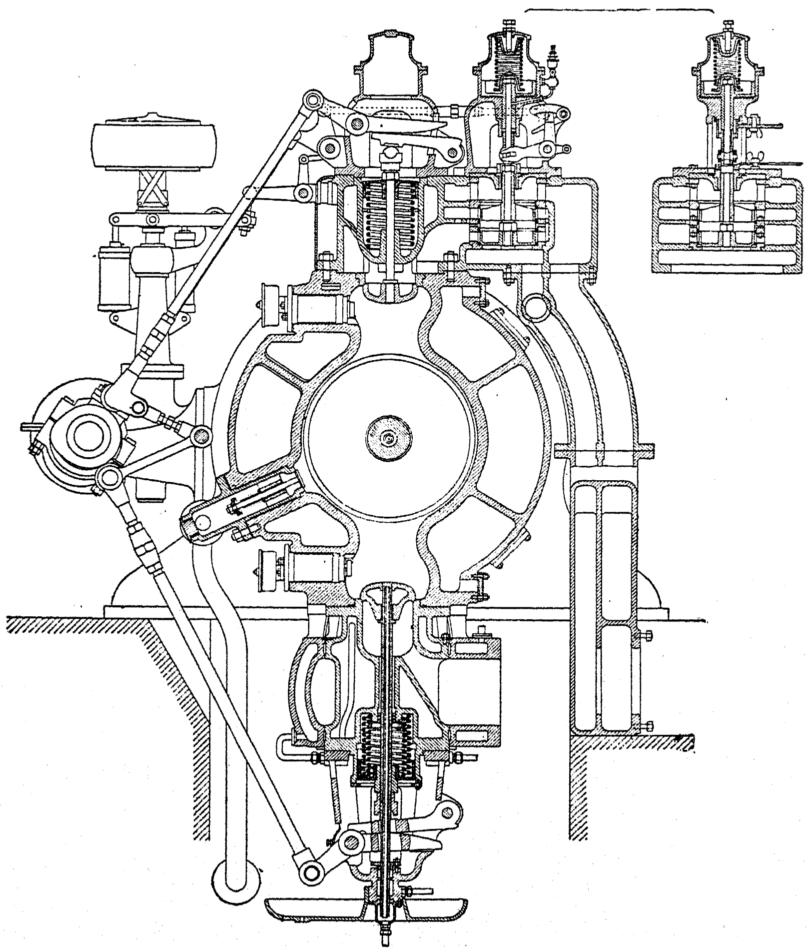


FIG. 36.—INLET AND OUTLET REGULATION OF THE DOUBLE-ACTING FOUR CYCLE ENGINE OF THE GUTEHOFFNUNGSHÜTTE, OBERHAUSEN.

anism actuated by the governor. For the admission of air and gas there are two circular slides on the same axis, so arranged that their openings can be separately regulated to obtain the best mixing during the working of the engine.

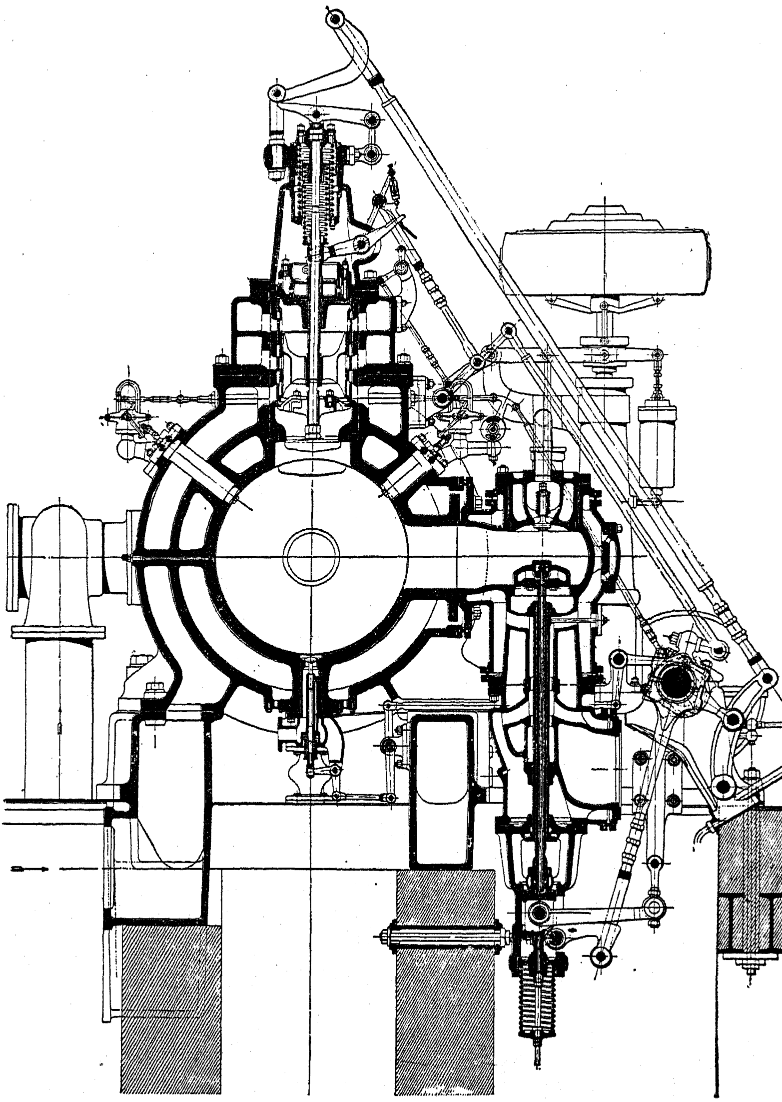


FIG. 37.—INLET AND OUTLET REGULATION WITH SIDE OUTLET-VALVE OF SCHÜCHTERMANN & KREMER, DORTMUND.

8. *Double-Acting Four-Cycle Engine by Schüchtermann & Kremer, Dortmund.* (Figs. 37, 38, and Plate VII.)

The Schüchtermann & Kremer engines, in the construction of which I have participated, differ from other types, principally by the outlet-valve being placed at the side of the cylinder in order to be more accessible, and by the previously described

system of governing for producing a constant quantity of mixture

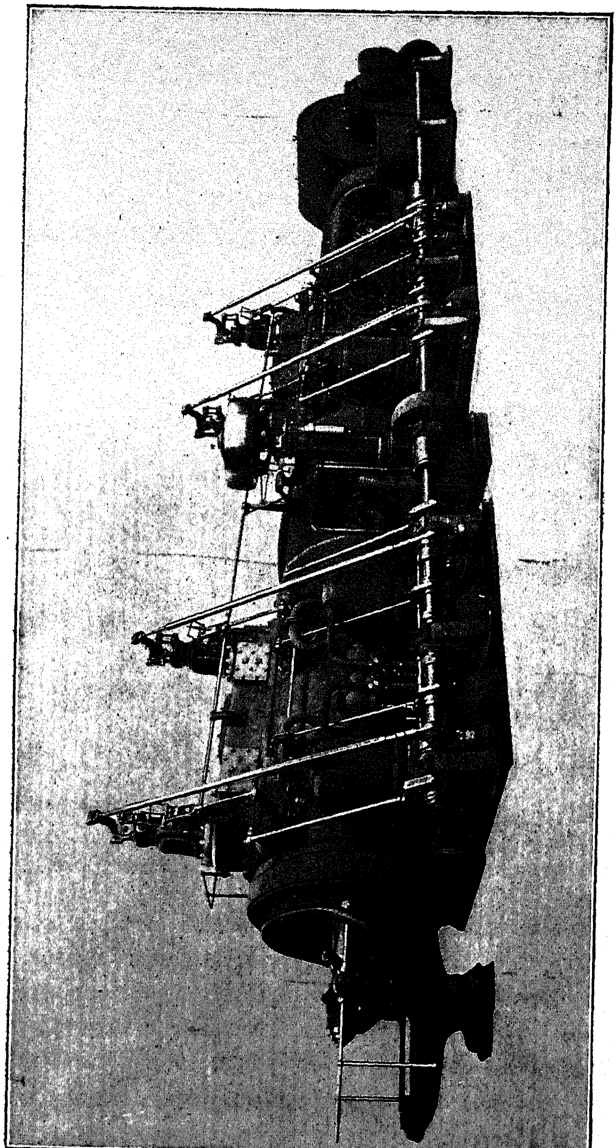


FIG. 38.—1,200-H.P. TANDEM GAS-ENGINE, BUILT FOR THE UNION AKT.-GES. FÜR BERGBAU, EISEN- UND STAHLINDUSTRIE BY SCHÜCHTERMANN & KREMER, DORTMUND.

and constant compression (Fig. 12). Both arrangements have given good results.

9. *Double-Acting Four-Cycle Engine by the Maschinenbau-Aktiengesellschaft Union, Essen-Ruhr.*

Plate VIII. and Figs. 39, 40 show the construction of this engine.

This construction is characterized by the Reichenbach valve-gear (Fig. 39), and also by the manner in which the cylinder is constructed.

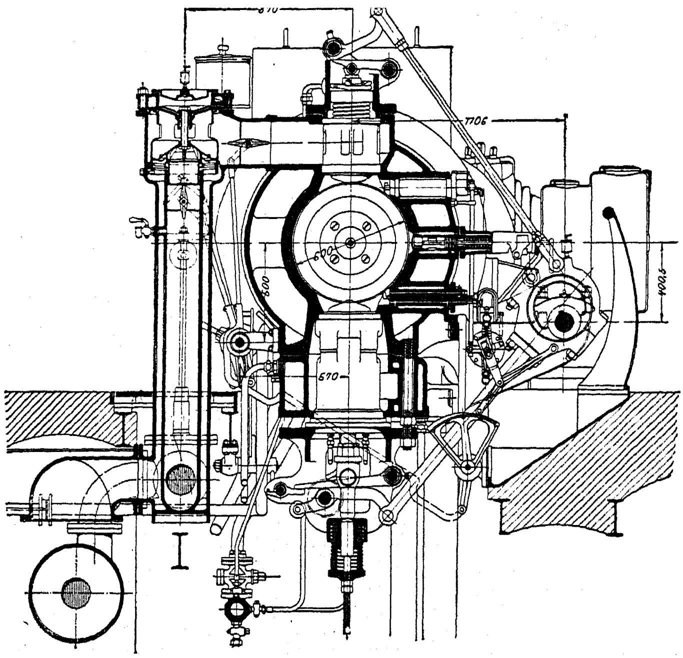


FIG. 39.—REGULATION OF THE REICHENBACH DESIGN GAS-ENGINE OF THE MASCHINENBAU-AKTIENGESELLSCHAFT UNION, ESSEN A. D. RUHR.

The outer jacket is partly open in the middle in a manner similar to that of the Gasmotorenfabrik Deutz engine, and covered by a plate cylinder; further, the jacket near the two end flanges is split after casting, and rendered tight by joints of rubber cords and wire.

Thereby conflicting stresses of the inner liner and of the outer jacket, caused by their different temperatures, at least in a longitudinal direction, are avoided, and also those of the flanges.

These flanges transfer the stress due to explosive action, by massive points, to the liner only.

One inlet-valve and the corresponding outlet-valve are operated by an eccentric with roller-levers. The inlet-valve is, in my opinion, unnecessarily cooled. It is stated by the makers that the hollow disk of the outlet-valve can be withdrawn through the cylinder by removing a screw. In large gas-engines the mixing-valve should not be automatic, but moved by the valve-gear shaft, and placed nearer to the inlet-valve in order to avoid a large reserve of mixture.

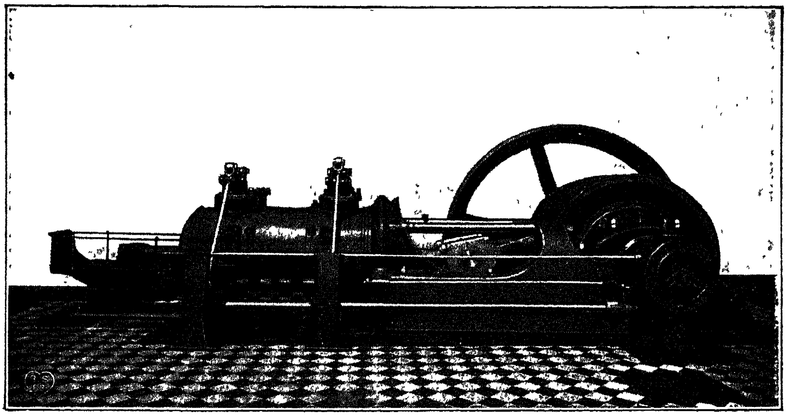


FIG. 40.—GAS-ENGINE OF THE MASCHINENBAU-AKTIENGESELLSCHAFT UNION, ESSEN A. D. RUHR.

10. *Double-Acting Four-Cycle Engine by the Duisburger Maschinenbau-Aktiengesellschaft, formerly Bechem & Keetman, Duisburg.* (Figs. 41, 42, and Plate IX.)

This engine is noteworthy in many ways. It has at the top of each end of the cylinder an inlet-valve and vertically under the same an outlet-valve, arranged in such a manner that the common axis of the valves is far enough to the side of the piston-rod to allow the outlet-valve with its spindle to be lifted up without hindrance when the inlet-valve and its seating has been removed (Fig. 41).

Further, a closed-link motion, automatic in action, with the working-piston as used in two-cycle engines, is employed, as well as an outlet-valve for the exhaust of the burnt gases (Fig. 42), so that, at the end of each explosion-stroke, the special piston composed of three parts first opens short slots, whereby the

pressure of the gases is equalized with that of the atmosphere, and only afterwards is the outlet-valve allowed to open. It is clear, without further comment, that by this means the outlet-valve is released before lifting, and that the exhaust-valve chamber and the piping connected therewith are no longer exposed to such high temperature.

It appears to me to be doubtful, however, whether, in the first place, the outlet-valve can really be made much smaller in the

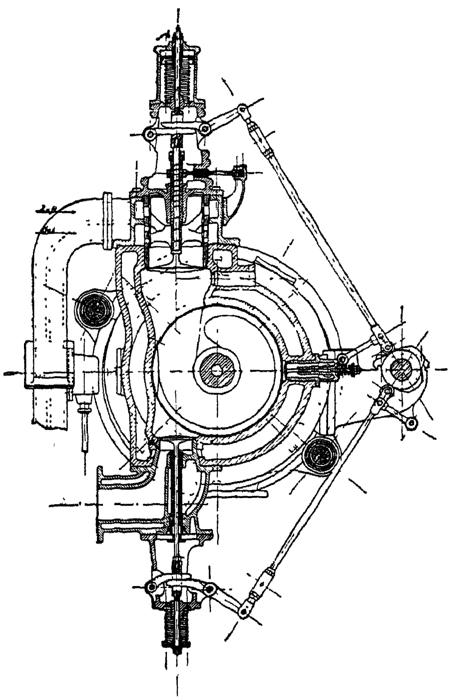


FIG. 41.—INLET AND OUTLET REGULATION OF THE DUISBURGER MASCHINENBAU-AKTIENGESELLSCHAFT, FORMERLY BECHEM & KEETMAN.

manner claimed by the builders, without causing a too high back-pressure during the whole length of the exhaust-stroke; and whether, secondly, it is not necessary, also, to cool the outlet-valves in larger engines; because if the gases passing through the outlet-valve are no longer so hot as with other four-cycle engines, yet this valve is situated in the explosion-chamber, without, as is the case with the inlet-valve, being cooled by the fresh entering mixture.

The distance between the outer piston-ends must be equal to

the stroke of the engine, and for this reason the engine must be considerably longer than the first described four-cycle engines.

The idea of exhaust slots or ports controlled by the piston in four-cycle engines is not new, since it has been taken into consideration by most builders of the newer type.

The valves are operated by cams with the aid of roller-levers, and the method of governing is by quality. The quantity method could not be employed, because with a light load, com-

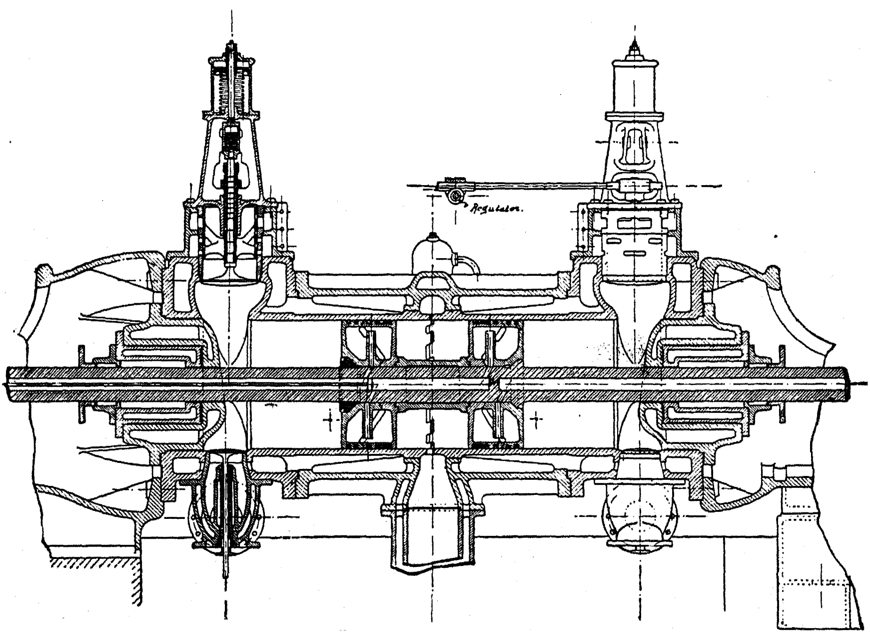


FIG. 42.—CYLINDER OF THE DOUBLE-ACTING FOUR-CYCLE ENGINE OF THE DUISBURGER MASCHINENBAU-AKTIENGESELLSCHAFT, FORMERLY BECHEM & KEETMAN.

bined with a corresponding back-pressure in the cylinder towards the end of the suction-stroke, it would cause too large a return flow from the exhaust-pipe through the slots.

According to the builders, the combustion, in spite of the quality governing, is complete at all loads. They attribute this to the peculiar form of the combustion-chamber, which should bring it about, that after ignition the portions of the charge that are not yet burning are put into motion and are directed into paths which lead to the portions which are already burning.

As may be seen from Fig. 42, the cylinder consists of three principal parts, an outer jacket strongly held in the middle, in which from both sides a liner, cast in one piece with the valve-chamber facings and a strong flange for bolting to the jacket, is inserted, so that the two cylinder-liners meet with a small clearance in the exhaust-slots. In the longitudinal axis of the jacket, for this reason, no tensile or compressive stresses can appear.

11. *Double-Acting Four-Cycle Engine by the Dingler'sche Maschinenfabrik A. G., Zweibrücken.* (Figs. 43, 44, 45, and Plate X.)

The Dingler construction differs from the former other modern four-cycle engines principally in the retaining of a cylinder open at one end. Double-acting is not arrived at in one cylinder, but properly in two single-acting cylinders, whose compression-chambers are bolted together (Fig. 43). Thereby the cylinder-ends, which contain the valves in their outer and inner casing, and as a continuation of thè latter, are cast together with the liner, and this is inserted into an outer jacket in such a manner that opposing stresses are avoided. Towards the crank-shaft bearing, this outer jacket continues in the form of crosshead guides and frame.

Both pistons are on the same piston-rod, which must traverse the cooled distance-piece between the compression-chambers through packings. This packing appears to me to be a very difficult detail of the Dingler engine. It is, of course, an advantage that the tightness of the working-piston can, at any time, be examined and adjusted, by reason of the cylinder being open; but this will be impossible as regards the tightness of the piston-rod. It will be seen from Fig. 43 that, at the moment in which the explosion takes place on one side of the distance-piece, the packing-rings are at the other end of the same, therefore the hot gases can penetrate to nearly the whole length of the bush. Consequently, the lubrication is very much affected.

The manner in which the piston is fastened to its rod is peculiar.

A split collar is provided with a series of projections on its inner surface. These projections engage similar parallel grooves that have been turned on the rod.

This collar is covered by a cylindrical sleeve, which has an outside flange to attach to the piston-face, and an inside flange to pull upon the collar. On screwing up the outside flange to

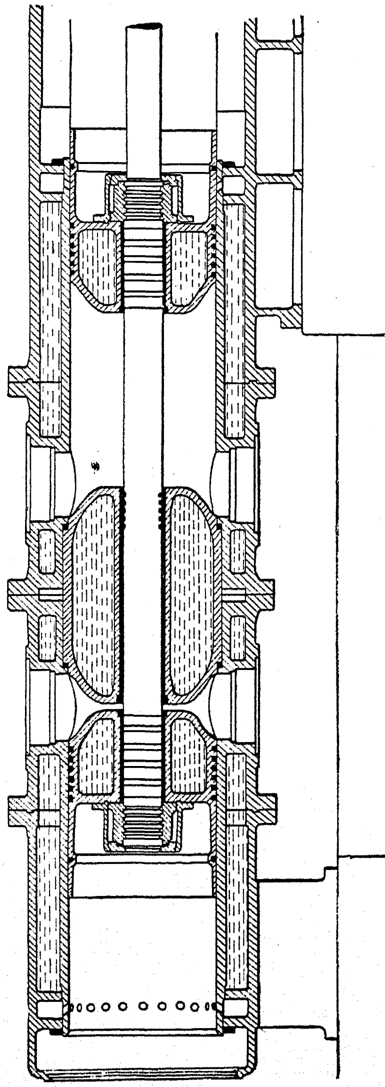


FIG. 43.—CYLINDER OF THE DOUBLE-ACTING FOUR-CYCLE GAS-ENGINE OF THE DINGLER'SCHE MASCHINENFABRIK, ZWEIBRÜCKEN.

the piston-face, the collar-projections tighten upon the back face of the grooves, thus putting the piston-rod in tension, and rigidly fixing it to the piston.

The advantage of this method of fastening is the facility with

which it can be disconnected. A number of spring rings are inserted between the rod and a bush placed in the boss of the piston, to make a gas-tight joint against the pressure in the cylinder. The valve-gearing actuates in each case an inlet-valve situated at the top and an outlet-valve situated at the bottom of the cylinder, whose motion is obtained from a common cam on the shaft, a. A second shaft, b, is placed in front of the shaft, a, which rotates with the same number of revolutions as the crank-shaft, and carries an adjustable regulating-cam controlled by a Dörfel flat governor. The action of the adjustable cam,

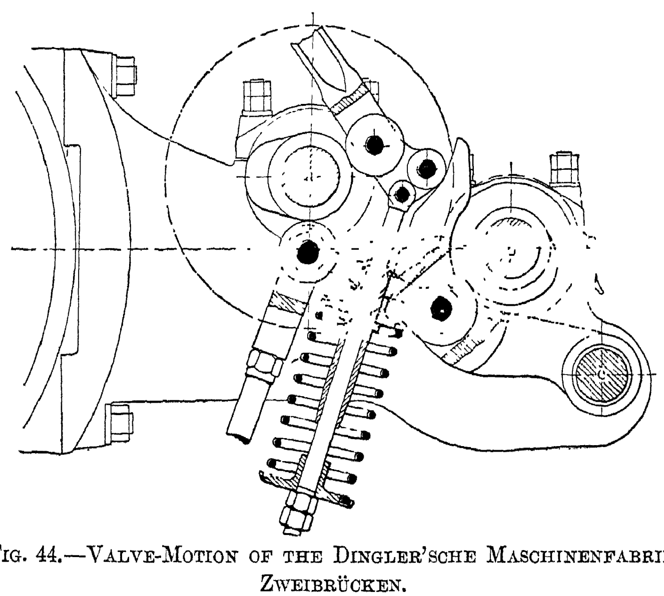


FIG. 44.—VALVE-MOTION OF THE DINGLER'SCHE MASCHINENFABRIK, ZWEIBRÜCKEN.

by a lever moving round the point, c, gives a very equal opening of the inlet-valve for all loads, while the lift and duration of the opening of the mixture-valve are variable. The governing is, therefore, a quantity governing with throttled mixture. The ignition is also adjusted by the governor.

The outlet-valve is not as accessible as it should be.

The engine constructed according to Dingler's arrangement is only slightly longer than double-acting engines with closed cylinders; its advantages consisting above all in the ease with which the piston is taken off.

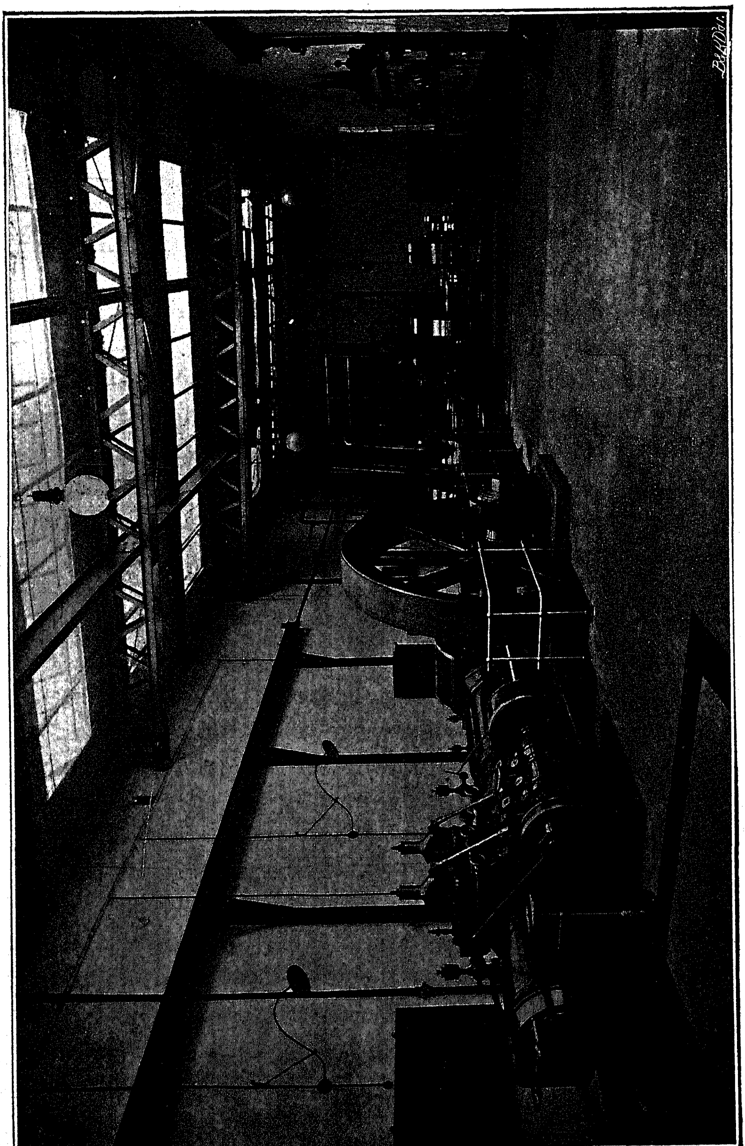


FIG. 45.—DOUBLE-ACTING FOUR-CYCLE GAS-ENGINE OF THE DINGLER'SCHE MASCHINENFABRIK, ZWEIBRÜCKEN.

In Germany the only two-cycle engines to be taken into consideration are the Oechelhäuser system and that of Körting.

The first is represented by the engines constructed by the Ascherslebener Maschinenbau-Akt.-Ges. and by A. Borsig, Berlin, the latter engines by Körting Brothers themselves and their concessionaires, who are: Gutehoffnungshütte, Oberhausen, Donnersmarckhütte, Zabrze, Siegener Maschinenbau-Aktiengesellschaft, and Maschinenbau-Aktiengesellschaft, formerly Klein Brothers, Dahlbruch.

12. *Two-Cycle Engine, Oechelhäuser System.* (Plates XI., XII., and Figs. 46, 47, 48.)

I assume that the manner of working and the advantages of these engines are known. They depend upon the use of the open cylinders and of inlet- and outlet-ports automatically controlled by the working-piston, avoiding the exposure of valve-heads and valves and also of stuffing-boxes and piston-rods to contact with fire; the balancing of the masses and the disappearance of the stress of the frame and foundation; so far a blowing-cylinder is not arranged tandem with the main cylinder. The design can be seen from the drawings of the Ascherslebener Maschinenbau-Aktiengesellschaft (Plate XI.) and of the A. Borsig engines (Plate XII.)

As compared with former examples of the Oechelhäuser engine, the principal modifications are to be found in the formation of the mixture and the manner of governing.

The charging-pump, which is usually placed behind the gas-cylinder, consists of a cylinder with automatic valves, the piston of which compresses, on the one side, gas, and on the other side air, to the necessary charging-pressure.

The scavening and charging of the working-cylinder take place during and shortly after the air-compression stroke of the charging-pump, so that air enters through the first open ports in the working-cylinder, and after the piston has overrun the gas-inlet ports an equalization of pressure in the charge-holders and pipes takes place and causes, at a definite moment, air and gas to enter together.

The mixture is thereby formed after the admission of air and gas in the cylinder itself.

The regulation of the speed—that is, the variation of the

charge corresponding to the load by the governor—is arranged differently by the two firms.

In engines by the Aschersleben Maschinenbau-A.G., the admission of gas only is regulated by the governor, and by means of König's patent return-valve, which puts the chamber situated around the gas-ports—that is, the pressure-vessel for the gas compressed in the charging-pump—in communication with the gas-suction pipe of the charging-pump, for a longer or shorter period at each revolution of the engine (Fig. 46).

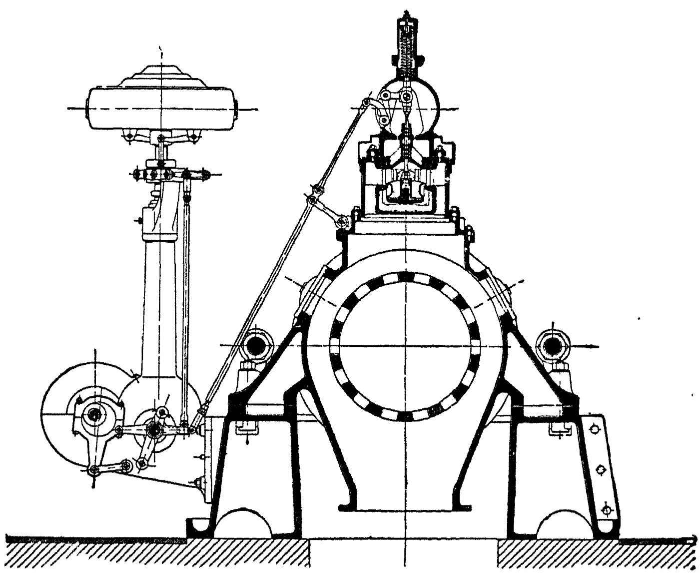


Fig. 46—König's Patent Automatic-Operating Return-Valve, Used by the Aschersleben Maschinenbau-Aktiengesellschaft.

For all loads the same quantity of air is employed, and the quantity of gas and the time of admission are varied according to the load.

If the return-valve is so regulated that when the gas-ports are opened by the working-piston, it is immediately closed, no gas or mixture can pass over in the gas-return pipe. During the simultaneous admission of gas and air, both are at the same pressure; nevertheless, only a variable mixture can be formed, because during formation the area of the air-inlet is constant, but that for the admission of the gas varies continuously. The governing of the Aschersleben engine resembles approximately

the quality method in four-cycle motors. It gives a variable mixture, and, when running without load, a weak mixture.

Nevertheless, according to information received from the firm, engines governed in this manner, which are coupled to alternating-current electric generators, can be connected in parallel without difficulty.

The method of governing employed by A. Borsig differs from that described above, in that: (1) The quantity both of gas and of air admitted to the working-cylinder is controlled by a return-flow valve, each valve being actuated by a Neuhaus-Hochwald gear. These valves are placed below the engine floor, and are situated in front of the engine, in the gas- and air-mains (Fig. 47). Further, the air admitted is divided by throttling into air for scavenging and for the mixture. (2) A circular valve or sleeve encircles the admission-passages (Fig. 47), which with a diminishing load, and also when the engine is running without load, gradually closes the openings situated opposite the point of ignition, so that when running without load there are only a few openings for the admission of gas where the ignition takes place. This circular valve is moved by gearing controlled by the governor, and represents an indirect method ot governing, the disadvantages of which are known, and in this connection could only be ascertained by experiment.

A. Borsig states that a simultaneous variation, in this manner, of the quantities of air and gas and of the areas of the inlet-passages was found to be necessary, otherwise the quantity of gas admitted to the cylinder in proportion to the quantity of air, when the engine was working almost without load, was so small that, when it was distributed throughout, the whole volume was incapable of forming a mixture strong enough to be ignited.

The experience gained by the Aschersle̱bener Maschinenbau-A. G. and by A. Borsig with reference to the governing is of such a contradictory nature that, to those unacquainted with the subject, it would appear to be unintelligible.

It must here be mentioned that in engines up to 1,000 effective h.p. in a single cylinder, A. Borsig arranges one charging-pump, which is placed in the axis of the working-cylinder, but, for powers above 1,000 h.p. in one cylinder, employs two double-

acting charging-pumps for gas and air, driven by one piston-rod, and situated below the floor of the engine-house.

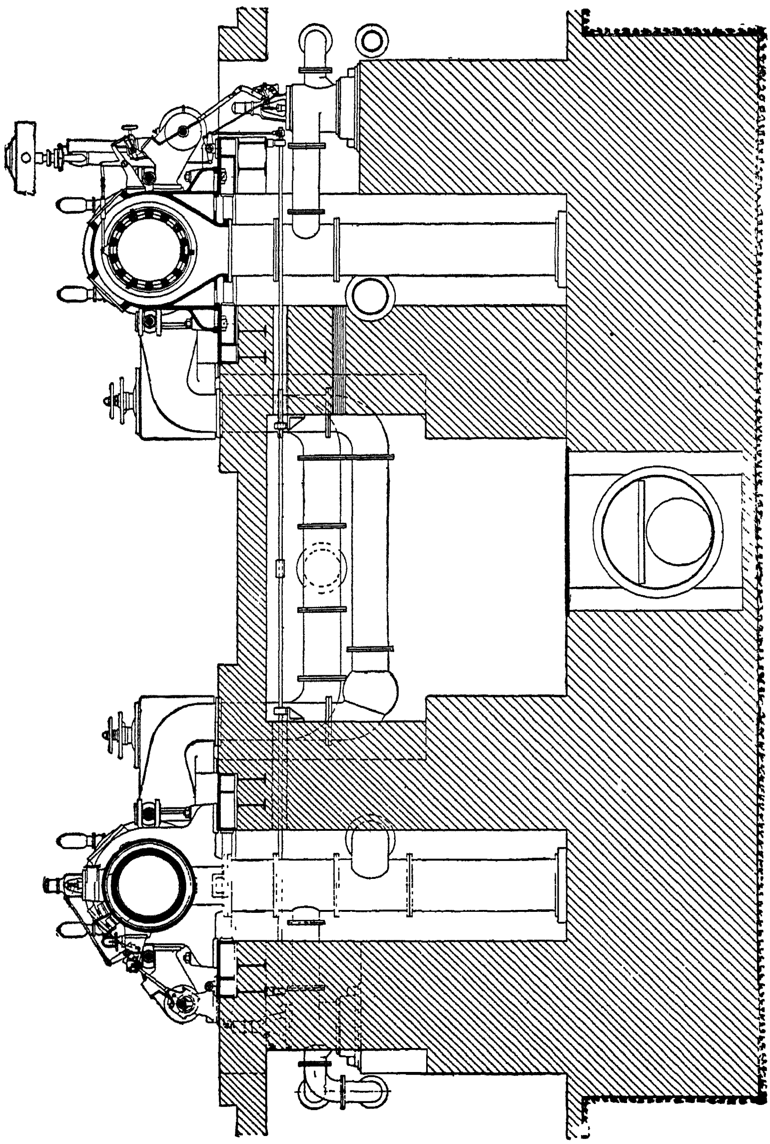


FIG. 47.—GOVERNING ARRANGEMENT OF THE GAS-ENGINE OF A. BORSIG, BERLIN.

As compared with former types, the Oechelhäuser engine as now constructed is much narrower, owing to the disappearance of the superfluous outside bearings for the fly-wheel.

13. *Double-Acting Two-Cycle Engine by Körting Brothers.* (Figs. 49 to 64, Plates XIII., XIV., and XV.)

The innovations which the above-named builders of Körting engines have introduced in the last few years, consist in the

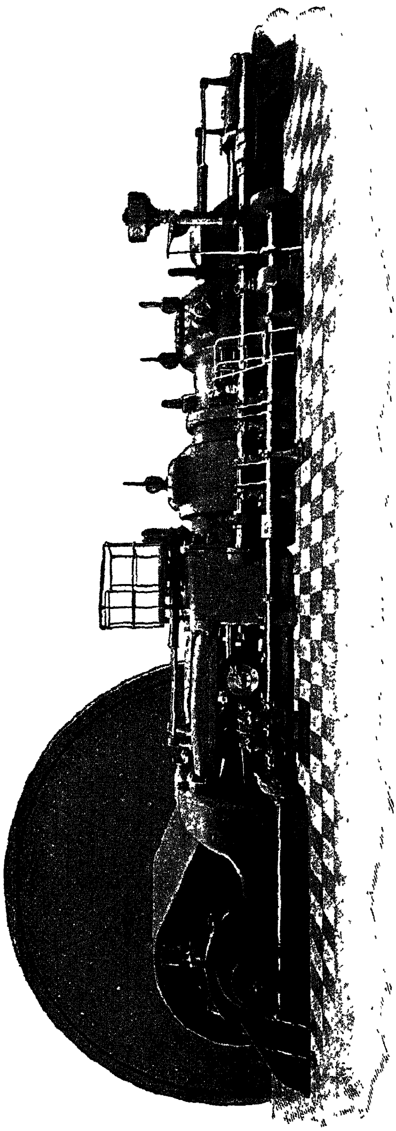


FIG. 48.—1,800-H.P. SINGLE-CYLINDER ENGINE OECHELHÄUSER SYSTEM, BY A. BORSIG, BERLIN.

satisfactory design of several parts, principally of the cylinder-heads and of the cylinder; also the simplification of the charging-pump, and improved methods of governing.

The double-acting in the Körting engine takes place on both sides of a piston, in a cylinder whose ends are closed by cylinder-heads. The piston automatically regulates the exhaust-ports, while the inlet of the air and the mixture takes place through an admission-valve situated in the upper portion of the cylinder-head. The cylinder itself stands on two side frames, which, continued forward, form the crank-shaft frames. Between these side-frames the flat crosshead-guide is arranged (Fig. 49, and Plate XV.).

Two separate double-acting charging-pumps, one for air and one for gas. are situated at the side of the working-cylinder.

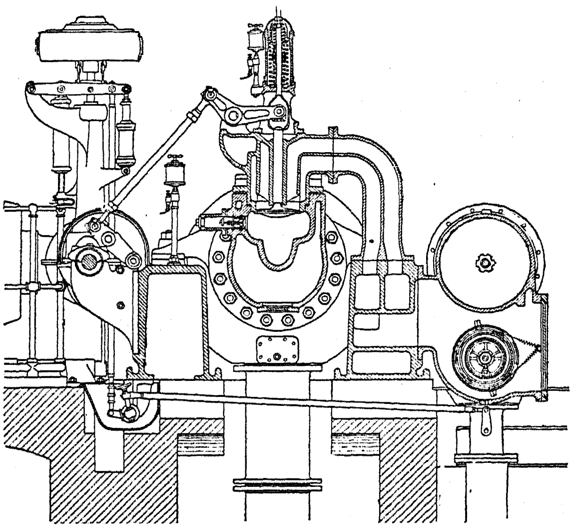


Fig. 49.—Cross-Section of a 400-h.p. Körting Engine by Körting Brothers, Körtingsdorf.

I have described elsewhere the principle on which these pumps work.[16]

From these pumps the air and the gas pressure-pipes lead into two concentric circular chambers above the inlet-valve, and as these chambers are constantly in communication with one another, the gas and air are always at the same pressure in the admission-passages.

According to the quantity of gas required, by the action of the governor, air, compressed to the charging-pressure, and always in equal quantity, enters from the circular air-chamber,

[16] *Stahl und Eisen*, vol. xxii., pp. 1157 to 1182 (1902).

and for some distance into the gas-passage, so that, at a definite moment depending on the load, when the inlet-valve is opened, air first, and then gas and air, enter the working-cylinder together at the same pressure, but in quantities proportional to the respective areas of the air- and gas-pistons of the charging-pumps, until the inlet-valve closes.

The formation of the mixture, as described, is ideal, and cannot be surpassed, for an entirely constant quantity is formed, with the proportion of gas and air as intended by the design,

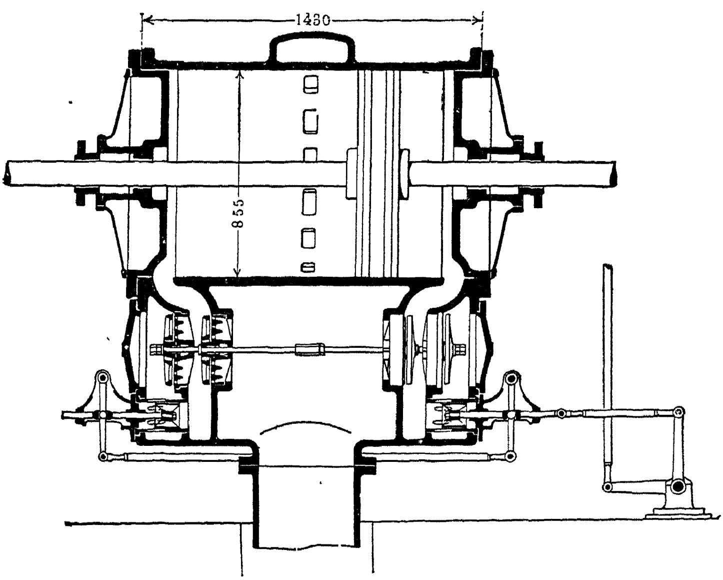


Fig. 50.—Gas-Pump, with Regulation, of the Maschinenbau-Aktiengesellschaft, formerly Klein Brothers, Dahlbruch.

and entirely independent of the pressure, or of the variation of pressure in the suction-pipes of the charging-pumps. In the Körting engines these variations of pressure, in reality, have no influence on the working of the motor, at least only to a slight degree at the time when the engine is working at full load.

At all loads and speeds of the engine a good mixture is always to be found, after compression, next to the ignition-point.

The compression varies with the load, while the quantity of air admitted is constant, and the quantity of gas is reduced with the load.

The inlet-valve is controlled by cams, and in most cases closed by springs. Considerable acceleration-resistances are caused by the rapid opening and closing of the valves, and for this reason the gearing for closing the valves should also be arranged with cams and roller-levers.

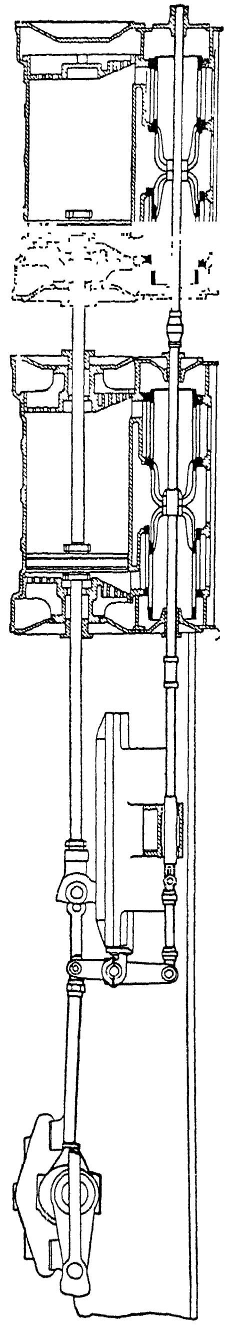


FIG. 51.—CHARGING-PUMP VALVE-MOTION OF THE SIEGENER MASCHINENBAU-AKTIEN-GESELLSCHAFT, SIEGEN.

To govern the speed of the engine—that is, to regulate the admission of gas, with corresponding loads—the Körting engine was originally fitted with a governor with a return-flow valve, in such a manner that a throttle-valve, controlled by the governor, allowed more or less gas to return to the cylinder of the charging-pump from the gas-pressure passage (between the cylinder and the charging-pumps), during the period of suction.

Though this system of governing makes extra work for the gas-pump, it is, even in the latter types of Körting engines, still employed, owing to its simplicity, as shown by the charging-pump by Klein Brothers (Fig. 50).

The charging-pumps were originally worked by an eccentric valve-motion with a circular valve, both for inlet and outlet.

In more modern engines the charging-pumps have direct-acting piston slide-valves for the inlet-valves and also for the outlet (Plate XV.), and by adjusting a double slide-valve motion, the quantity of gas can, at the same time, be regulated.

The charging-pumps manufactured by Klein Brothers and the Siegener Maschinenbau-Aktien-Gesellschaft are much more simple. The first have no eccentric valve-

motion, but only automatic valves (Fig. 50). In this pump Klein Brothers have taken advantage of the peculiarity of the Körting engine—namely, that the air-pump compresses during the complete stroke and the gas-pump commences to compress in the middle of the compression-stroke. For this reason the cylinder of the gas-pump is provided in its middle portion with ports, which communicate with the gas-suction. The gas can return to the suction-pipe through these ports during the first half of the compression-stroke. This regulation of the charging-pump avoids the unnecessary consumption of oil and power that obtains in the former slide-valve motion.

The charging-pump gearing of the Siegener Maschinenbau-Aktien-Gesellschaft is shown in Fig. 51. The eccentric motion is here retained; but a single eccentric with a slide-valve rod moves all the inlet-valves of the gas- and air-pumps. The exhaust-valves of the charging-pumps are placed in the cylinder-covers.

FIG. 52.—GAS-PUMP REGULATION DIAGRAM OF THE SIEGENER MASCHINENBAU-AKTIEN-GESELLSCHAFT, SIEGEN.

The valves situated under the pump-cylinders are provided with tapered openings, and work in casings with similar openings. When the gas-slide is moved by the action of the governor, the tapered openings of the slide and of the casing are brought nearer to, or farther from, each other. Thereby at the beginning of the suction period a variable "after opening" of 0 to 10 per cent. takes place, and the suction-pipe is closed during the pressure-stroke (after an admission of from 35 to 80 per cent.). A diagram from this charging-pump is reproduced in Fig. 52.

The air-pump can be regulated in a similar manner by hand. This simple motion avoids the return of compressed gases.

With clean gas this motion could probably be actuated by the governor; but if the gas is not very clean, and at the same time is wet, the friction of the throttle-slide of the type of Klein Brothers, and, at times, the resistance of the inlet-valve of the type of the Siegener Company, will be excessive.

In newer Körting engines the liners of the working-cylinders and the jackets are no longer cast in one piece; they are more frequently made in halves, inserted into the jacket, so that the outer and inner cylinder-liners can expand independently (Fig. 53).

All builders of the Körting engine have retained the cylinder-heads bolted to the cylinder, and from the modifications, as shown in Fig. 54, as compared to the usual constructions, it may be concluded that they gave trouble by cracking.

The newer designs of the cylinder-heads avoid the pressure of the stuffing-box chamber on the outer wall of the cylinder-head.

The long pistons are cast in one piece, and have either a boss extending throughout their whole length or a short boss at

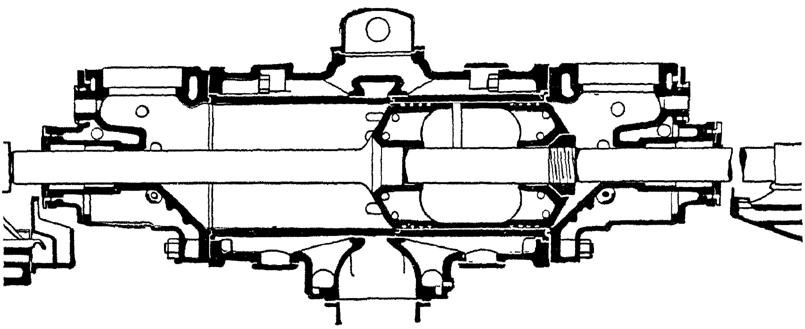


FIG. 53.—WORKING-CYLINDER OF KÖRTING ENGINE OF THE MASCHINENBAU-AKTIENGESELLSCHAFT, FORMERLY KLEIN BROTHERS, DAHLBRUCH.

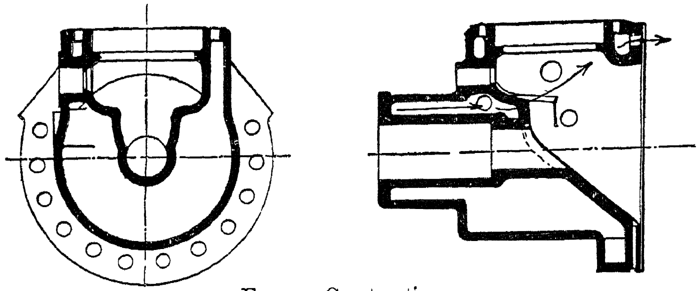
each end. In large engines the pistons are supported by back or front guides; in smaller engines the guide is dispensed with, and in its place the middle of the under surface of the piston is lined with anti-friction metal (Fig. 55).

Fig. 56 shows how the piston can be removed. It will also be noted that the piston-rod can be readily disconnected from the crosshead (Fig. 57).

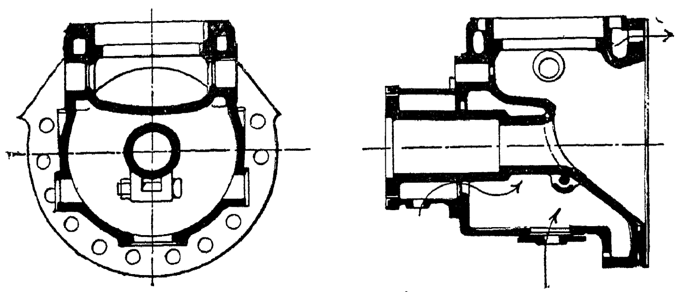
The Körting engine is specially suited for driving blowing-cylinders, because it starts easily when loaded, and is certain in its working at great variations of speed. and moreover when the speed is very low.

Fig. 58 shows the arrangement of the valve-gearing of the Corliss inlet-valves of a blowing-engine constructed by the

Siegener Maschinenbau A.G. In this gear, by means of a link with releasing mechanism, on the one hand, the amount of the



Former Construction.



Present Construction.

FIG. 54.—CYLINDER-HEAD OF THE KÖRTING ENGINE, AS CONSTRUCTED BY THE DONNERSMARCKHÜTTE, ZABRZE O. S.

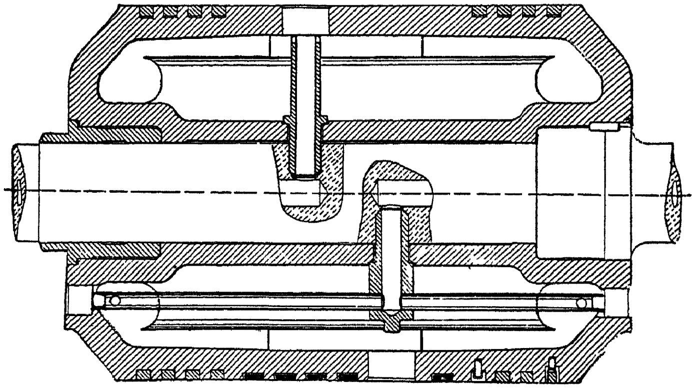


FIG. 55.—PISTON OF KÖRTING ENGINE WITH WHITE-METAL BUSHING, AS CONSTRUCTED BY THE GUTEHOFFNUNGSHÜTTE, OBERHAUSEN.

lap of the valves, and, on the other hand, the stroke and the advance of the eccentric, can be varied.

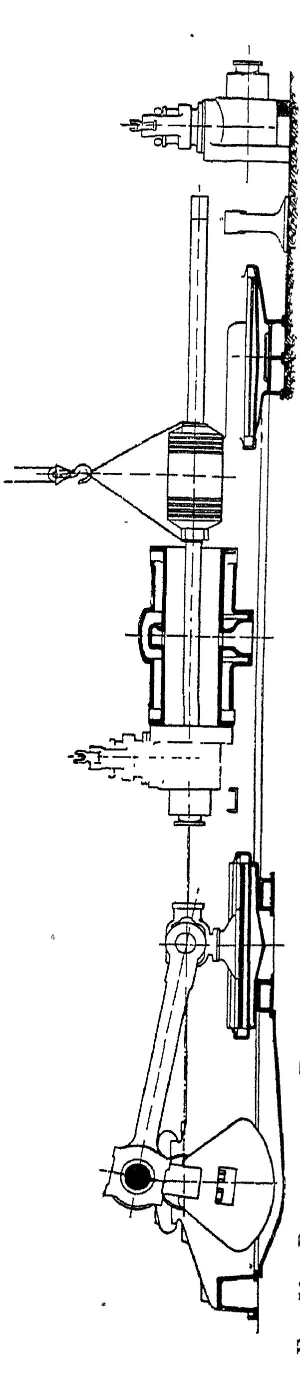


Fig. 56.—Removing the Piston of a Körting Engine by the Siegener Maschinenbau-Aktien-Gesellschaft, Siegen.

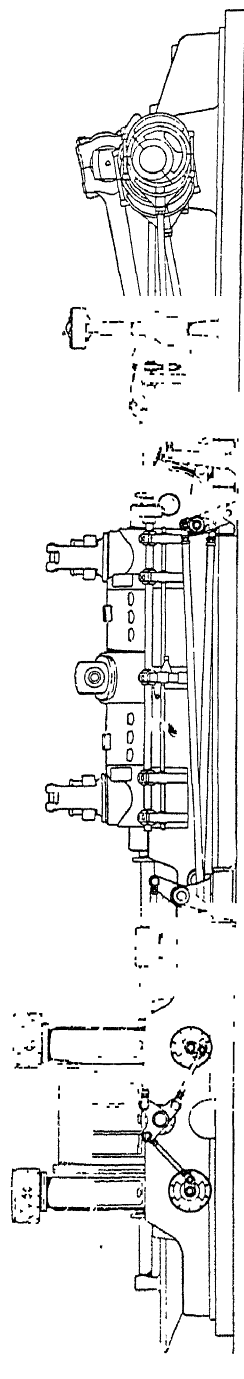


Fig. 58.—Arrangement of Suction Air-Valve in the Blowing-Engine of the Siegener Maschinenbau-Aktien-Gesellschaft, Siegen.

By this means, with an almost constant opening for the admission, the closing of the inlet-valve is brought about, at the commencement of the pressure-stroke at the ordinary load, but when the load is taken off, at the end of the pressure-stroke. Between these limits the variable admission of the blowing-cylinder is arranged for blowing smaller quantities of air at a higher pressure with approximately an equal expenditure of power.

The same object is attained by other builders by arranging return-flow valves on the blowing-cylinder. In this connection, for example, Klein Brothers make the valves in such a manner that they can be controlled by hydraulic pressure from the

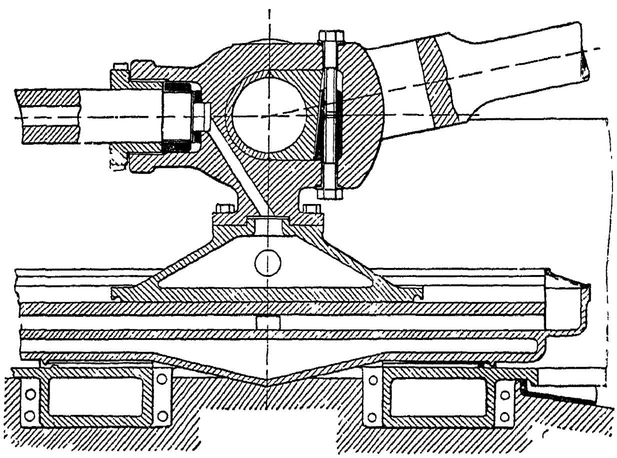


FIG. 57.—FASTENING THE PISTON-ROD AT THE CROSSHEAD IN THE KÖRTING ENGINE OF THE SIEGENER MASCHINENBAU-AKTIEN-GESELLSCHAFT, SIEGEN.

central position occupied by the driver, so that, as with the Siegener arrangement, if a blast-furnace scaffolds, or other similar accidents occur, the blast can be suddenly shut off.

I wish to remark here that most of the iron-works have answered my question as to the most practical size of the engines, viz.: for driving dynamos, from 1,000 to 1,200 b.h.p. per unit; for driving blowing-cylinders, for each blast-furnace a separate engine, usually, say, from 1,000 to 1,200 b.h.p., or in some cases larger units, say from 1,600 to 3,600 b.h.p., according to the efficiency of the furnace. Concerning the size of the engines, it may be observed that with a few large units, as compared with a larger number of smaller units, the reserve power of the

plant, also the safety of the apparatus—for example, of the pistons and cylinder-covers—diminishes; and further, that the cleaning of the large engines is more complicated and takes more time.

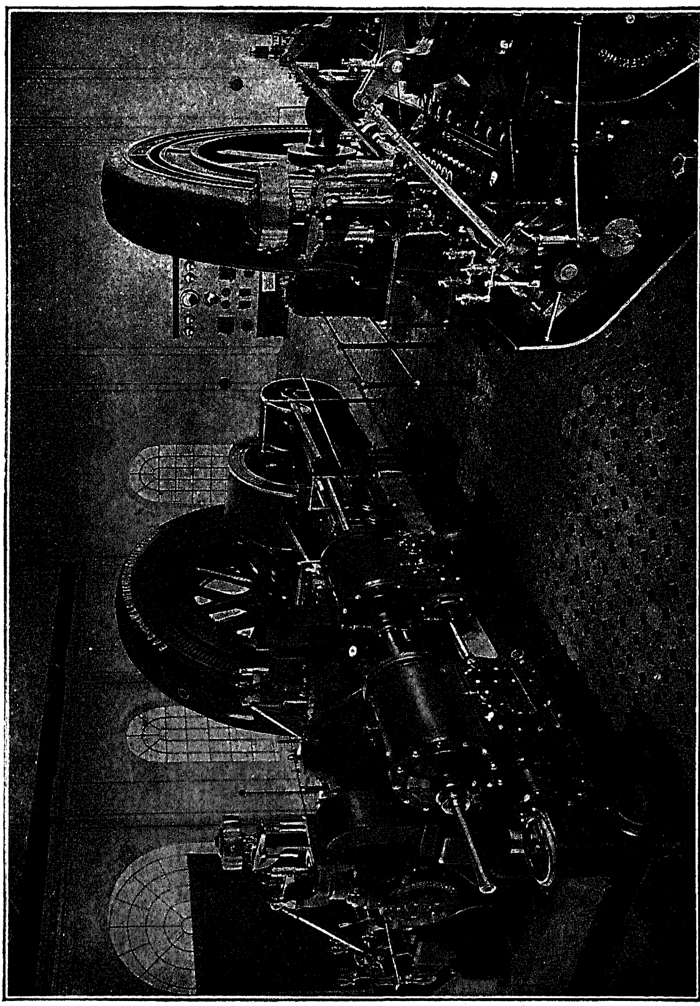


FIG. 59.—GAS-DYNAMO, BUILT FOR GRUBE MESSEL, BY KÖRTING BROTHERS, KÖRTINGSDORF.

Units larger than 1,000 to 1,200 effective h.p. are only to be found in very large plants, or where the space is limited.

From the answers received to my last question concerning the possibility of connecting alternating-current dynamos in parallel, when driven by gas-engines, this, stated generally, can

be done without special difficulty. From several sources it is gathered that sometimes the practice of switching in, at the

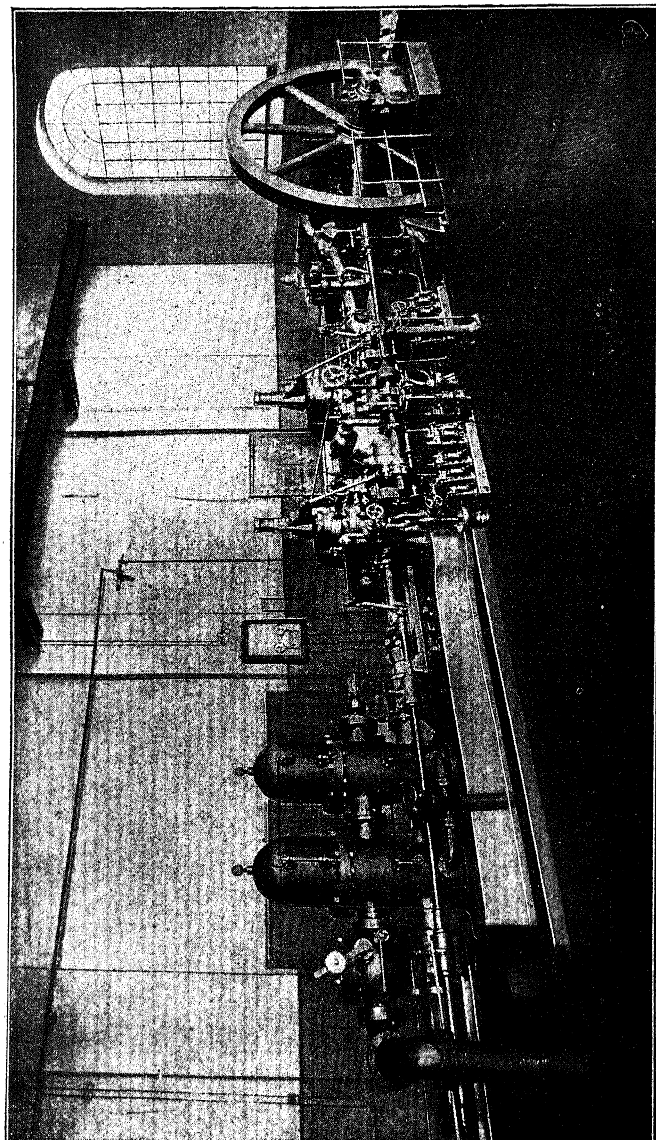


FIG. 60.—ACCUMULATOR-PUMP DRIVEN BY KÖRTING GAS-ENGINE, BUILT FOR THE HÖRDER BERGWERKS- UND HÜTTENVEREIN BY THE SIEGENER MASCHINENBAU-AKTIEN-GESELLSCHAFT, SIEGEN.

moment that the engine is running without load, occasions difficulty; that naturally depends not only upon the degree of uni-

formity of the engine, on the momentum of oscillation of fly-wheel, and on the construction of the dynamo, but above all

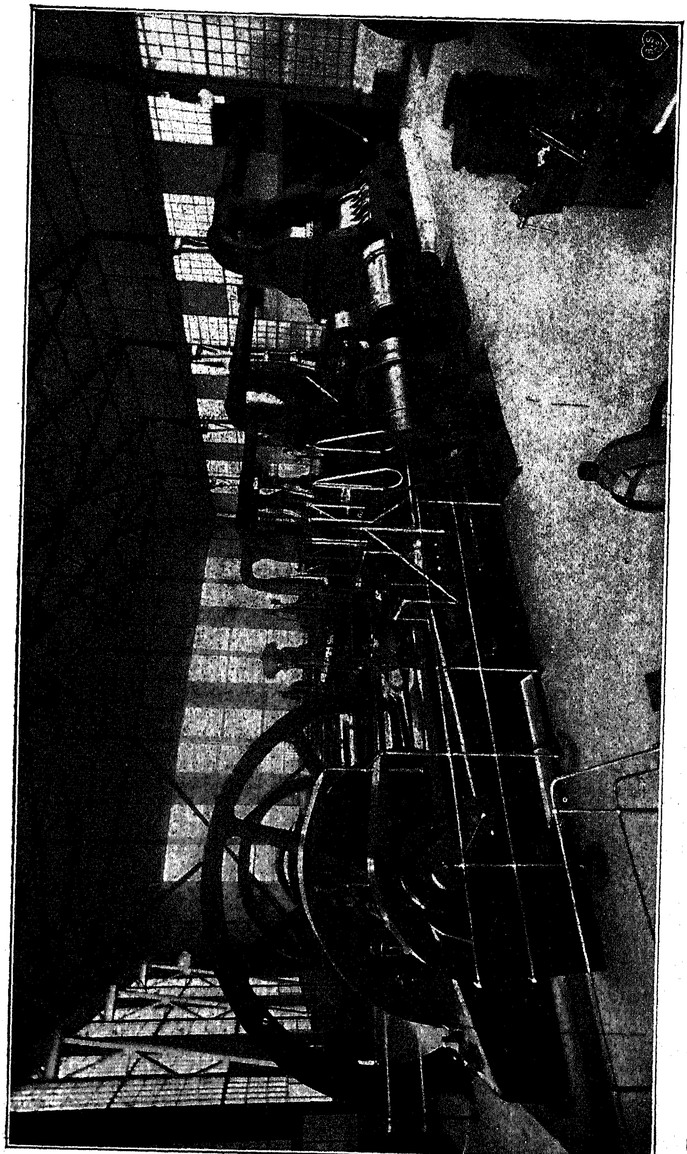


Fig. 61.—Twin Gas-Driven Blowing-Engine with Körting Motor, Built for Fried. Krupp Akt.-Ges., Rheinhausen, by the Siegener Maschinenbau-Aktien-Gesellschaft, Siegen.

upon the action of the governor and the formation of the mixture at the time named, and for this reason will vary with the design of valve-gear.

After considering all these various types, regarding the question as to which system—two-cycle, or double-acting four-cycle—should be adopted, I wish to make the following statement:

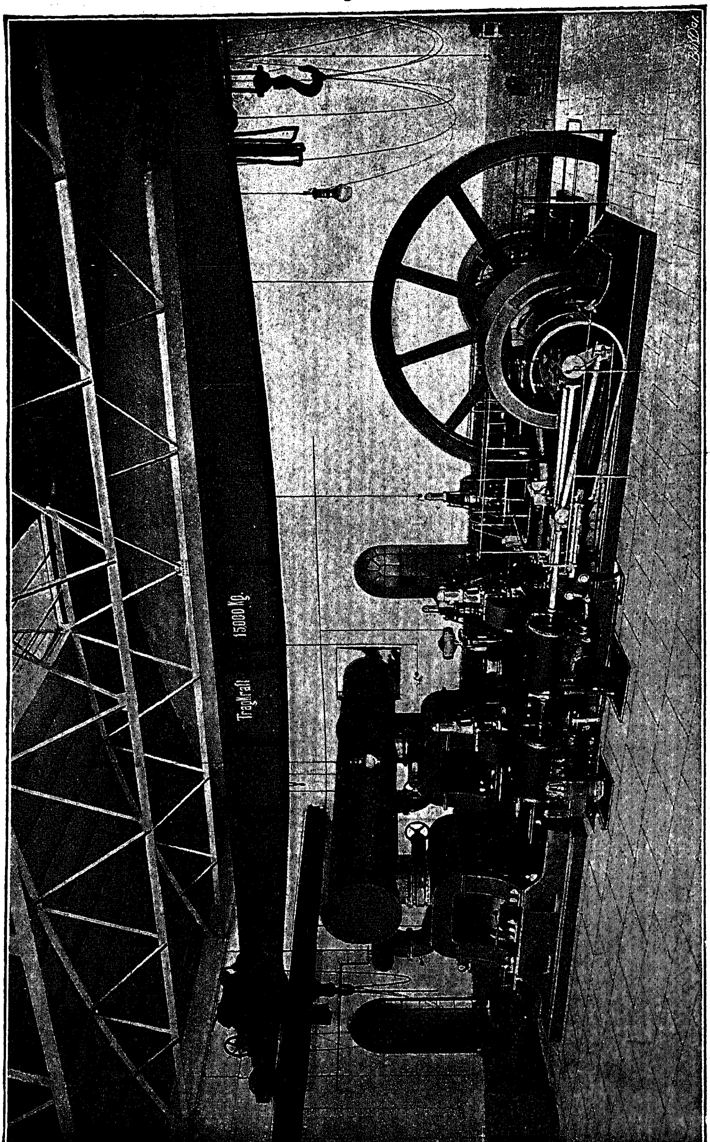


Fig. 62.—Gas-Driven Blowing-Engine, Built for the Gutehoffnungshütte, Oberhausen, by Körting Brothers, Körtingsdorf.

When the double-acting two-cycle engine was introduced by Körting Brothers in the year 1902, a number of engines made by them met with general success; this engine, as compared

with the then-existing single-acting four-cycle engines, showed such marked progress in this line that many considered that

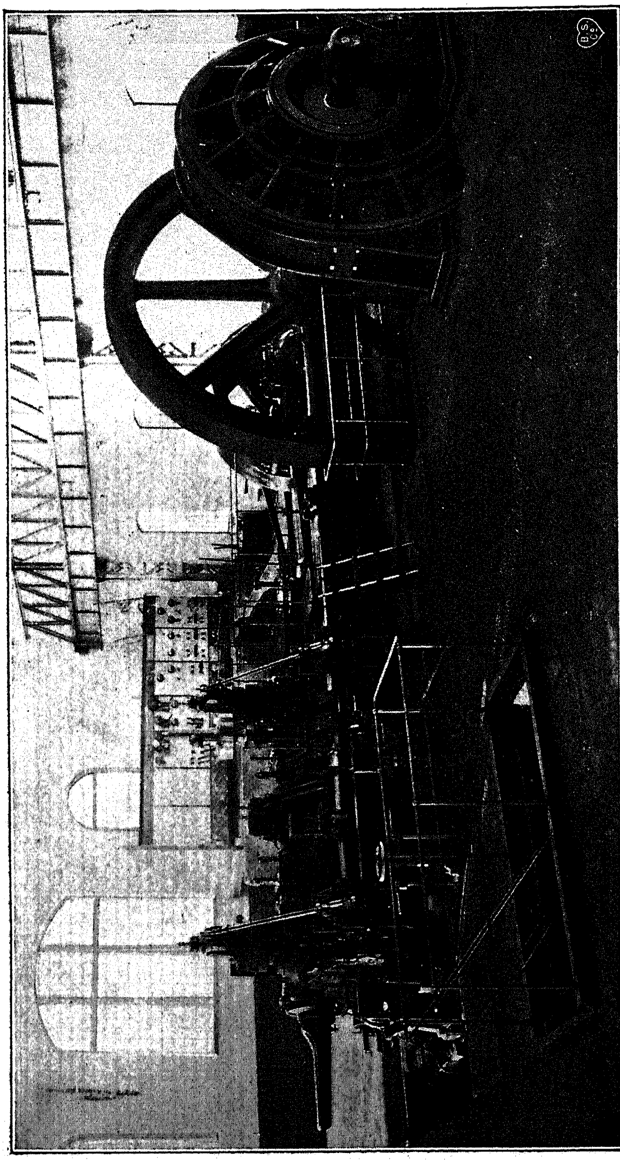


Fig. 63.—Körting Gas-Dynamo, Built for the Eisen- und Stahlwerk Hösch, Dortmund, by the Maschinenbau-Aktiengesellschaft, formerly Klein Brothers, Dahlbruch.

the four-cycle system was no longer a serious competitor. The builders of four-cycle motors were, however, guided by the success of the Körting engines in the right direction—namely,

of reconstructing their engines as double-acting engines in a single closed cylinder, and to arrange two of these cylinders, the one behind the other, with the object of increasing the me-

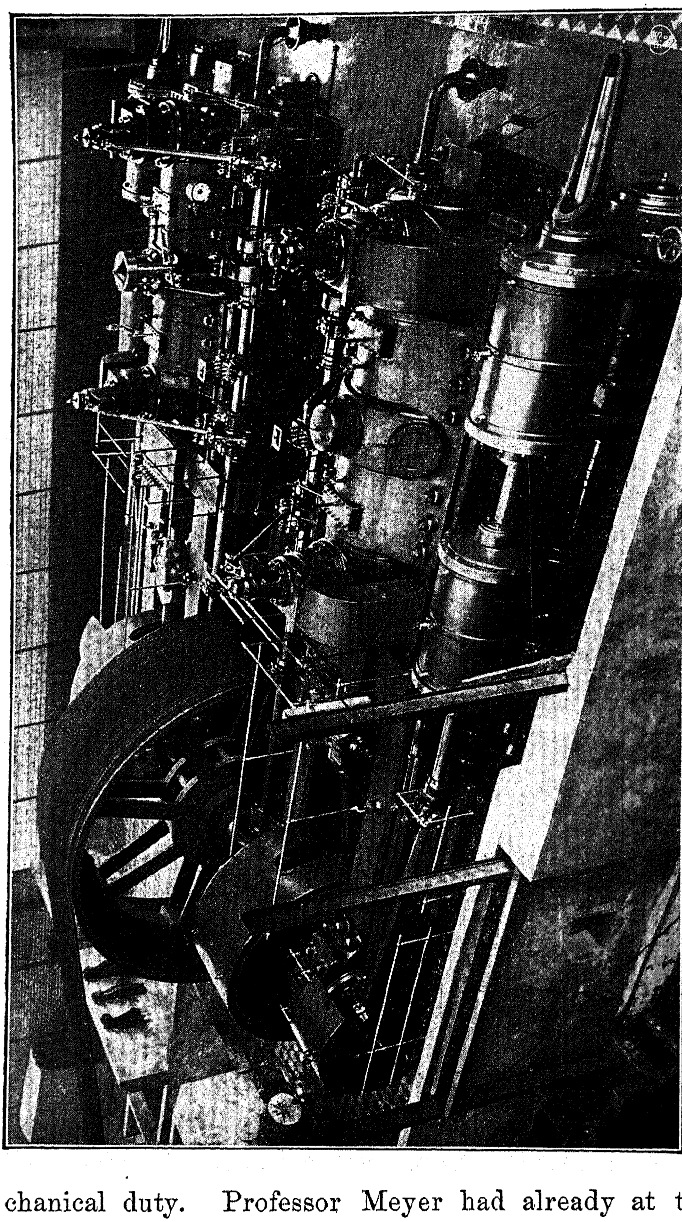


FIG. 64.—TWIN KÖRTING ENGINE OF ABOUT 1,800-H.P. FOR DRIVING A WIRE-TRAIN IN DIFFERDINGEN, BY THE MASCHINENBAU-AKTIENGESELLSCHAFT, FORMERLY KLEIN BROTHERS, DAHLBRUCH.

chanical duty. Professor Meyer had already at that time called attention to the fact that, in view of the large negative work expended in the charging-pump of the Körting engines,

the double-acting tandem four-cycle engine would, without doubt, again seriously compete with the Körting engine. That this forecast was a correct one, may be seen from the statistics given at the commencement of this paper.

Many persons at the present time, in judging the question of system, go so far as to prophesy that "the gas-engine will return to its original starting-point—viz., the four-cycle system—while others, equally convinced, affirm that the four-cycle will not continue to be adopted by iron-works and other manufactories."[17] Even with the experience gained up to the present time, it is impossible to give an opinion, based upon sufficiently trustworthy evidence, in favor of either of these opposing views. It is impossible to come to a conclusion from the total horse-power without further considering the proportionate value of the two systems; for up to March of this year, engines with 260,000 b.h.p. were at work, or on order, for double-acting four-cycle as against 91,000 for two-cycle. If these figures prove that the competition of the two-cycle engine must not be neglected, on the other hand, the importance and connection of the builders of the four-cycle types must be considered advantageous to the latter.

The builders of two-cycle engines will themselves admit that these motors are less adapted for the high speeds required for driving dynamos than for driving blowing-engines and pumps, because, with a reduced time of charging, the resistance of the charging-pump cannot be kept low enough; and principally, owing to the relatively larger number of explosions (especially with gases of high calorific value, such as coke-oven gas), the flow of heat through the metal walls, and thereby affecting the security of the explosion-chamber against breakage and the occurrence of premature ignitions, create uncertainty; as, moreover, the governing hitherto employed in two-cycle engines for dynamo-driving is, as a rule, inferior to that of the four-cycle engines. For this reason it may be explained that several firms who build two-cycle engines have lately decided to adopt also the manufacture of double-acting four-cycle engines.

On the other hand, the two-cycle engine is, without any

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[17] Güldner, *Entwerfen und Berechnen der Verbrennungsmotoren*, 2nd edition, p. 190.

doubt, most suitable for driving blowing-cylinders; for, as already stated, it permits, within wide limits, a variation in the number of revolutions per minute; it starts easily against a load, and at the low speeds of the blowing-piston the work of the charging-pump is not excessive. Klein Brothers, for instance, state that the work of their charging-pump with valves is from 6 to 7 per cent. of the work of the power-cylinder, so that the difference, compared with the negative work of the four-cycle motor, no longer preponderates.

Theoretical discussions concerning the correct or the incorrect mechanical efficiency, which during last year created such a stir in Germany, can for the present contribute nothing to elucidate the question of the systems. For the managers of works, in addition to inquiring about the price and power of a gas-engine, above all inquire about the security in working, and least of all about the quantity of gas consumed per b.h.p. They do not trouble themselves at all about the mechanical efficiency.

Suitable trials concerning the consumption of gas in more recent engines are not available for comparison, therefore it is not known how far the two-cycle engine is at the present time, in this respect, inferior to the four-cycle engine. Should the iron-works now be compelled to consider an economy of gas, I do not believe that the larger consumption of the two-cycle engines would for long have any great influence on the question of systems; for then the iron-works would probably first consider a more thorough cleaning of the gas employed for heating the blast and for burning under the boilers, thereby increasing its value, and in this manner saving gas. So long as these conditions remain as they are, and so long as the four-cycle engine is not more secure, under average working conditions, than the two-cycle engine, so long will the question of systems not be decided by general and theoretical considerations, but in such cases by the iron-works and mining industries themselves.

The situation was well summed up by a manager who, referring to this very point, told me personally that he himself preferred double-acting four-cycle, but his engine-drivers preferred double-acting two-cycle.

As regards other industries, they need not at present be con-

sidered to such an extent that they could assist in deciding the question.

In conclusion, it is my duty to state that the present position of the application of gas-engines in German iron-works shows the value the managers of these undertakings attribute to the better and less dangerous utilization of the waste gases of their furnaces, and the successful efforts that have been made by the German engineers in order to meet the requirements suddenly demanded by iron-works. It also proves that German iron-works are obliged to utilize to the uttermost the sources of power at their disposal, in order to insure their existence, and their participation in the trade of the markets of the world. In other richer countries, with more favorable conditions, the matter is not so urgent.

PLATES.

Secretary's Note.—Plates I. to XV. in the pamphlet edition published by the Iron and Steel Institute were printed on individual inset folders. In the arrangement of the paper for the *Transactions*, these illustrations have been placed more conveniently by extension over two opposite pages.—R. W. R.

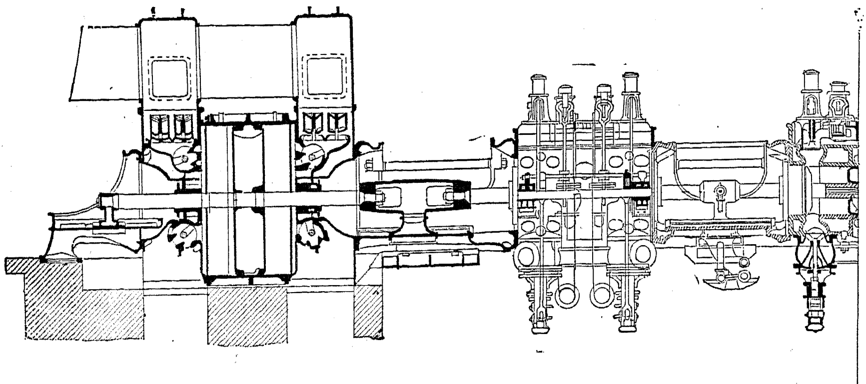


PLATE I.—GAS-DRIVEN BLOWING-ENGINE, BY

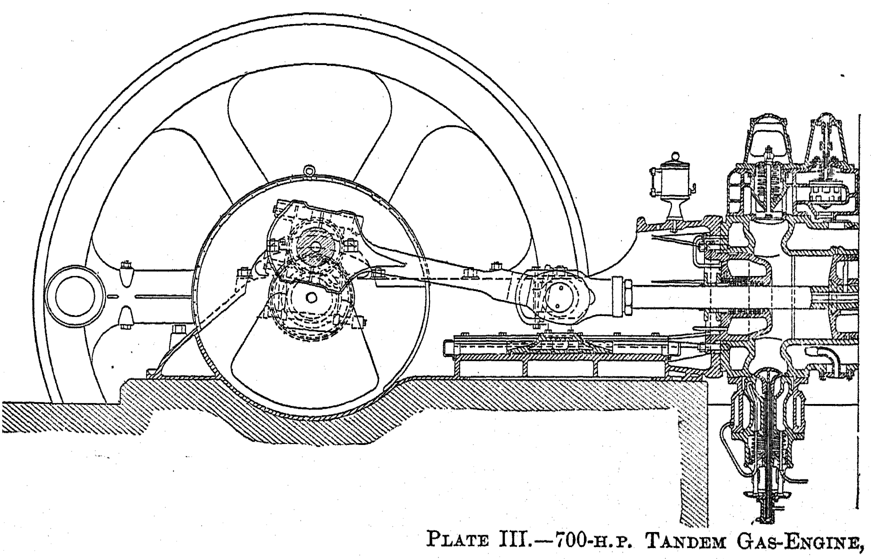
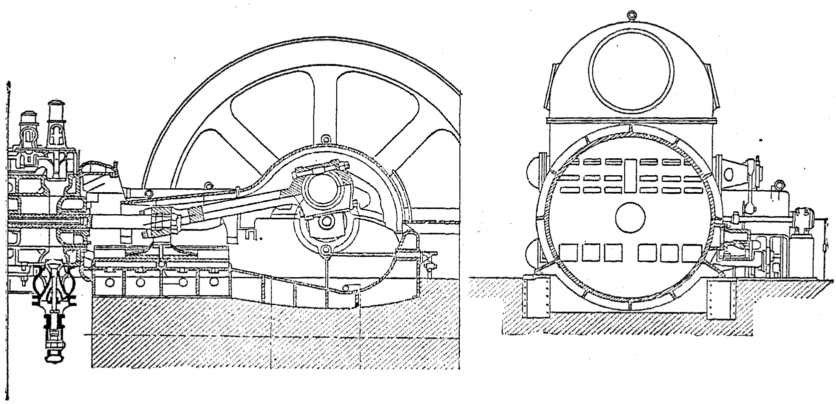
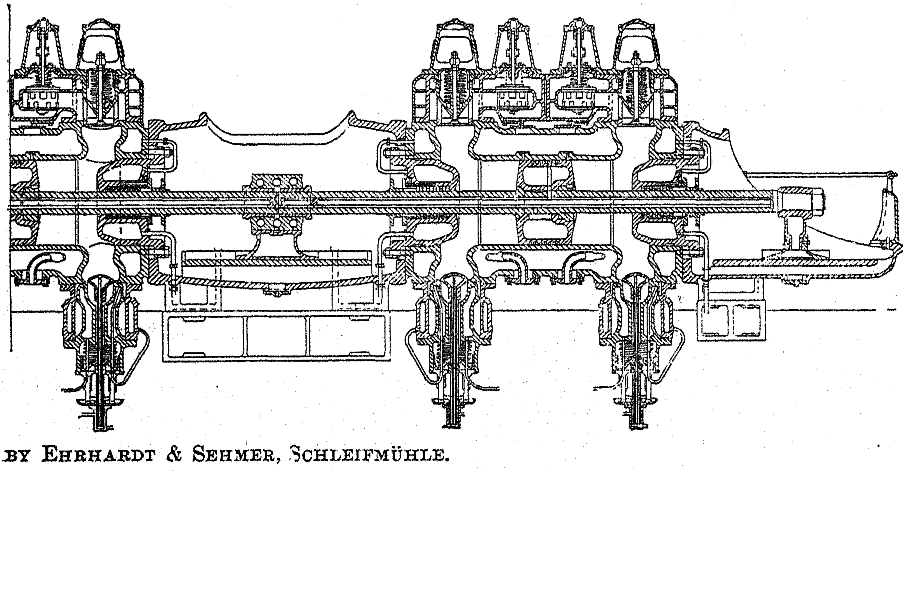


PLATE III.—700-H.P. TANDEM GAS-ENGINE,



THE MASCHINENBAU-GESELLSCHAFT NÜRNBERG.



BY EHRHARDT & SEHMER, SCHLEIFMÜHLE.

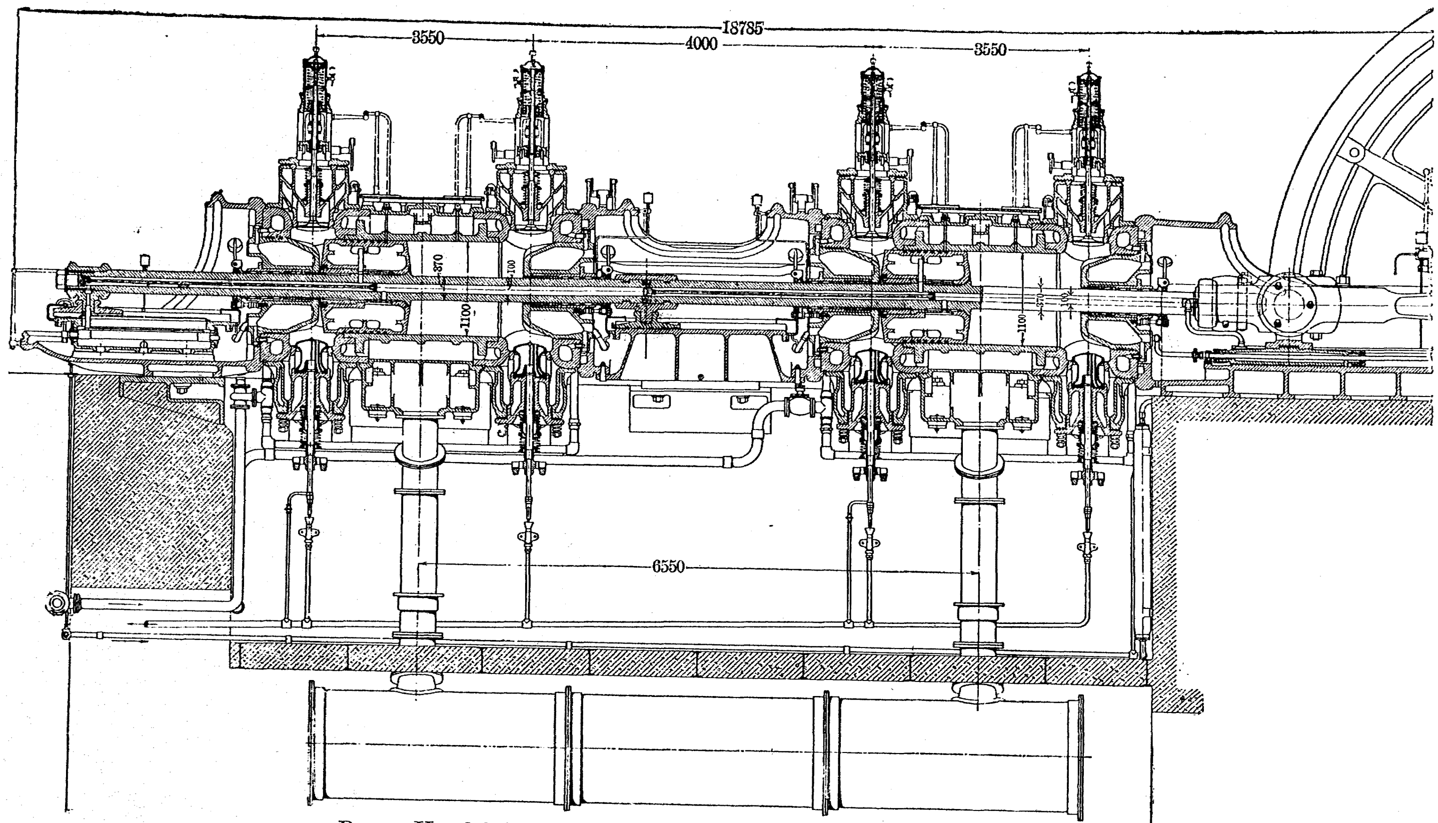


PLATE II.—2,000-H.P. GAS-ENGINE, BY THE GASMOTORENFABRIK DEUTZ.

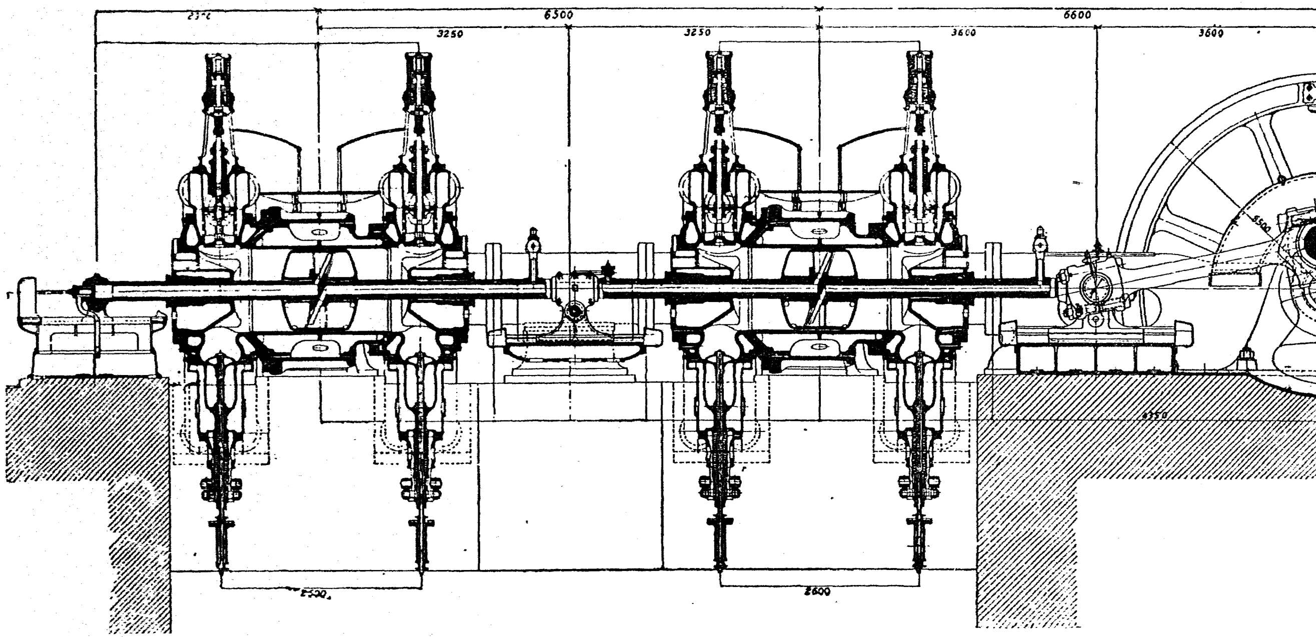


PLATE IV.—1,500-H.P. TANDEM GAS-ENGINE, BY THE MÄRKISCHE MASCHINENBAU-ANSTALT, WETTER A. D. RUHR.
(See also the two following pages.)

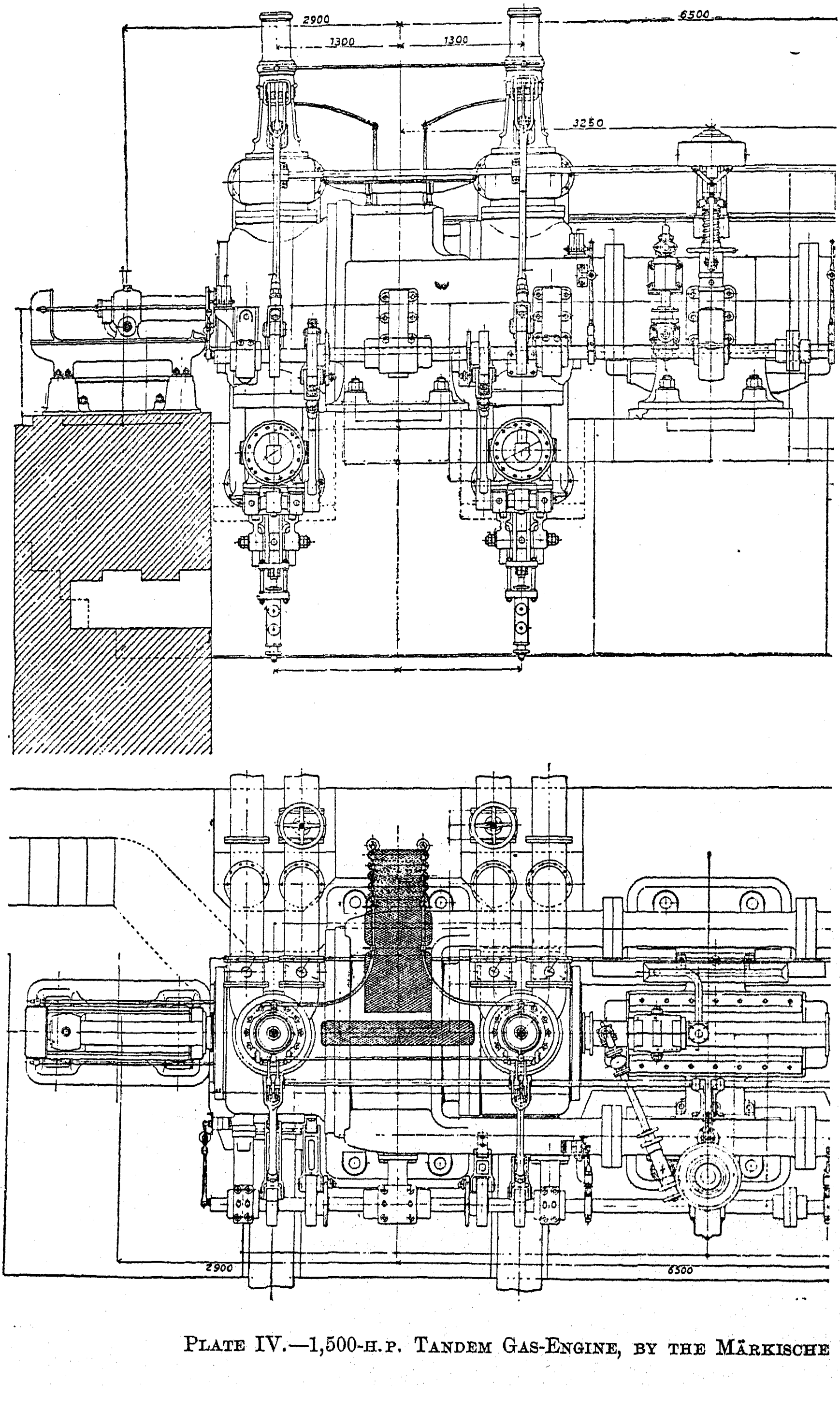
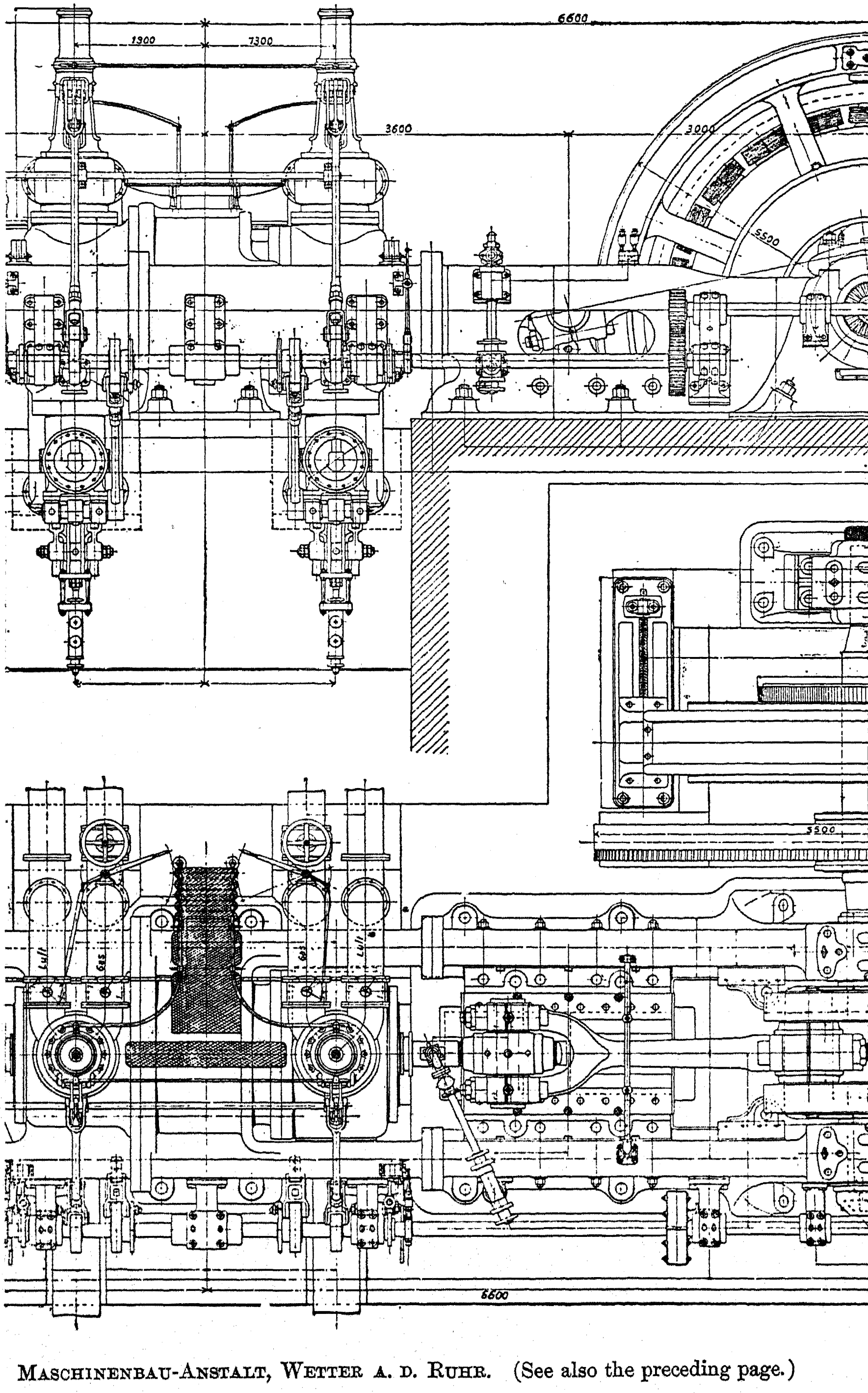


PLATE IV.—1,500-H.P. TANDEM GAS-ENGINE, BY THE MÄRKISCHE



MASCHINENBAU-ANSTALT, WETTER A. D. RUHR. (See also the preceding page.)

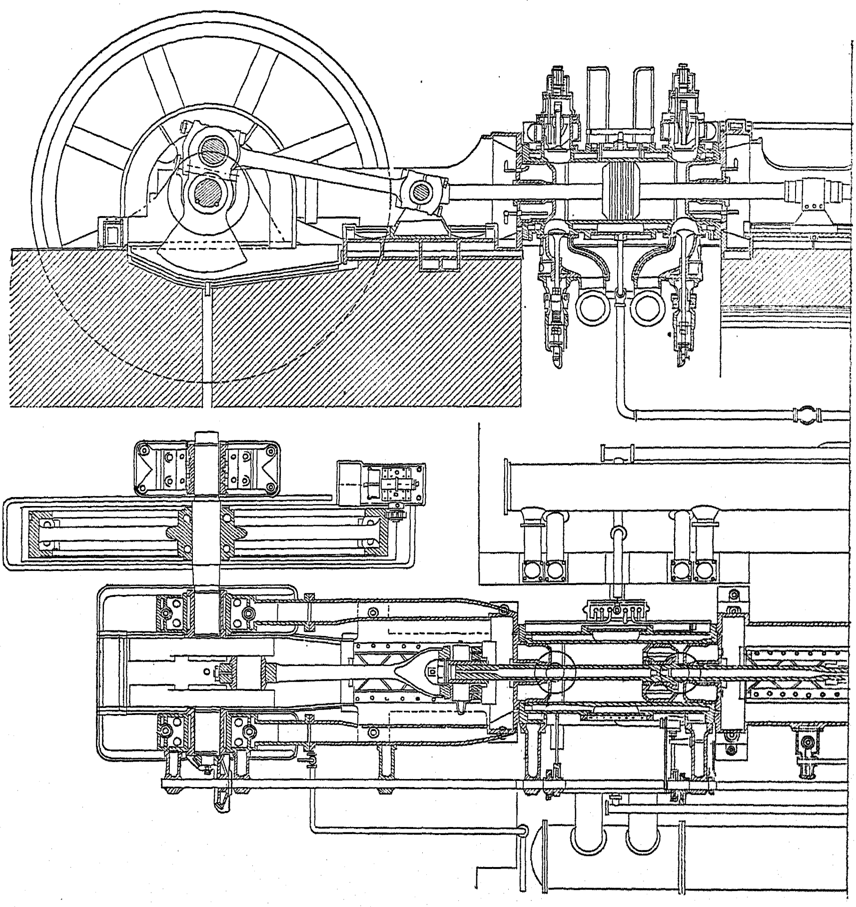
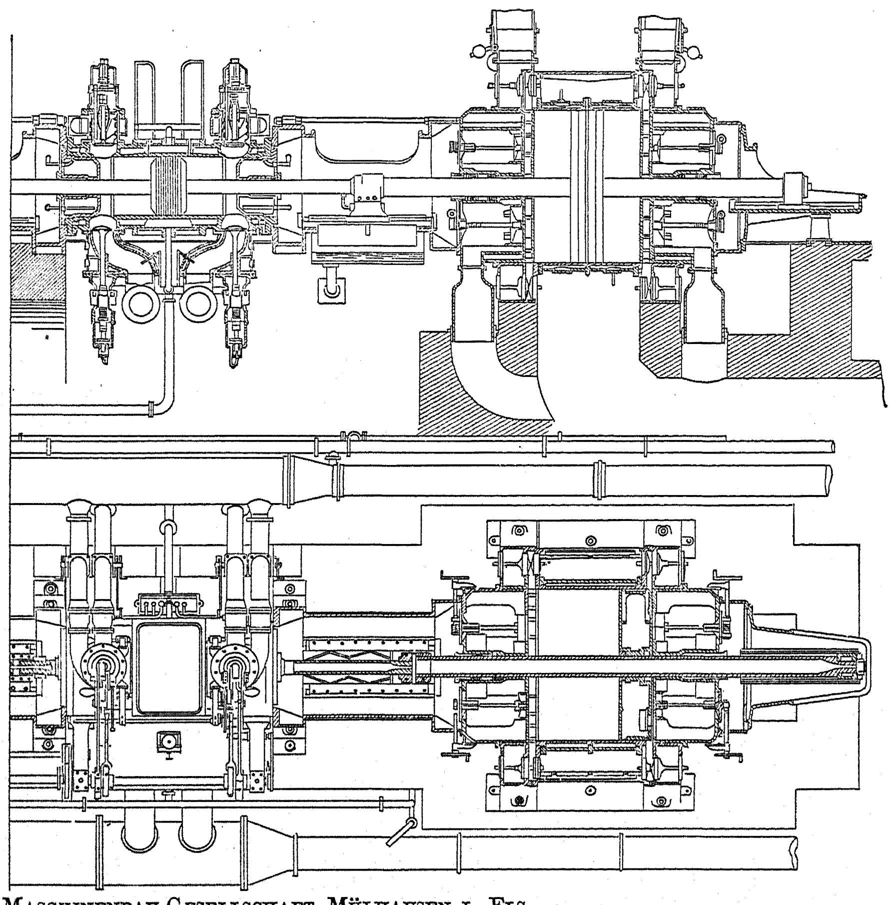


PLATE V.—1,500-H.P. BLOWING GAS-ENGINE, BY THE ELSÄSSISCHE



Maschinenbau-Gesellschaft, Mülhausen i. Els.

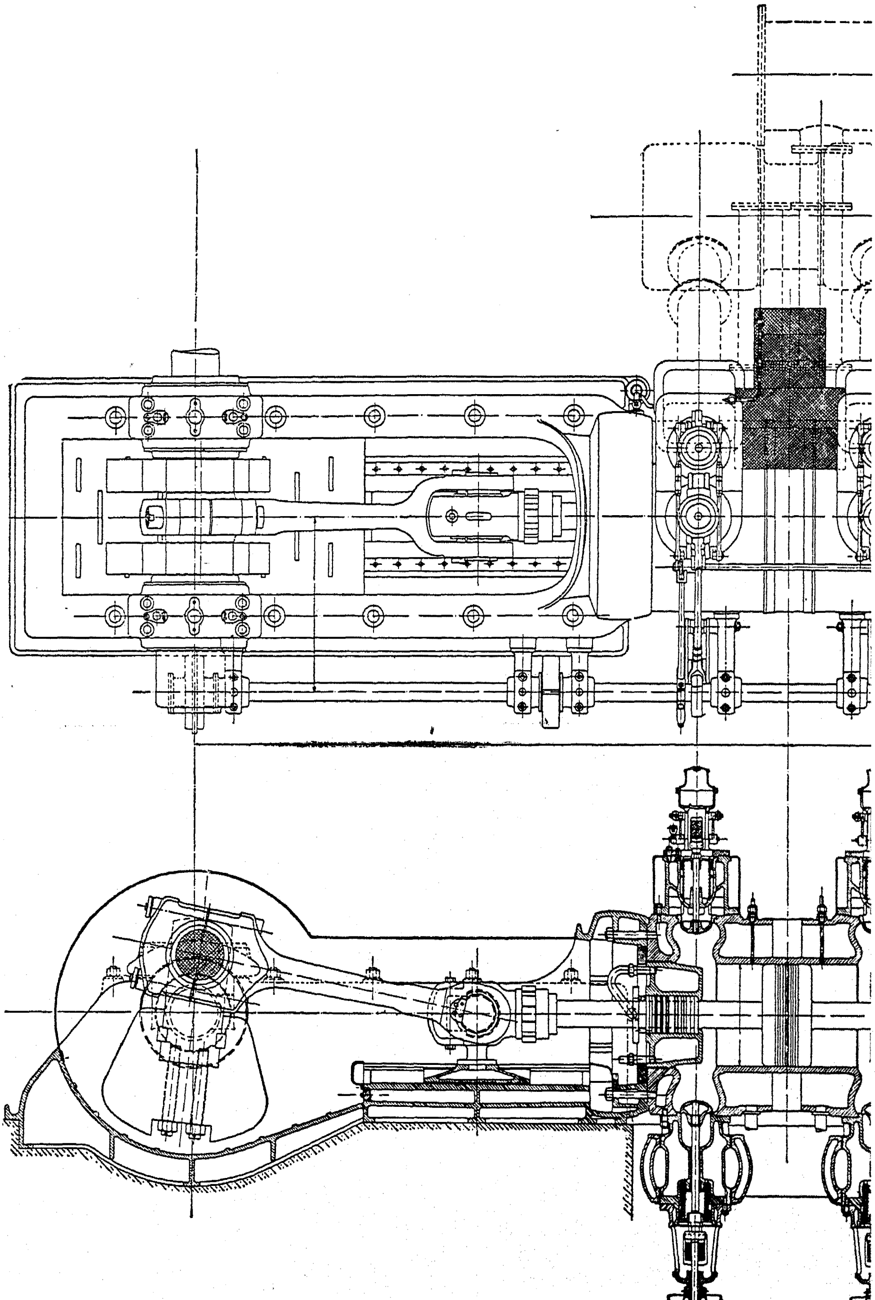
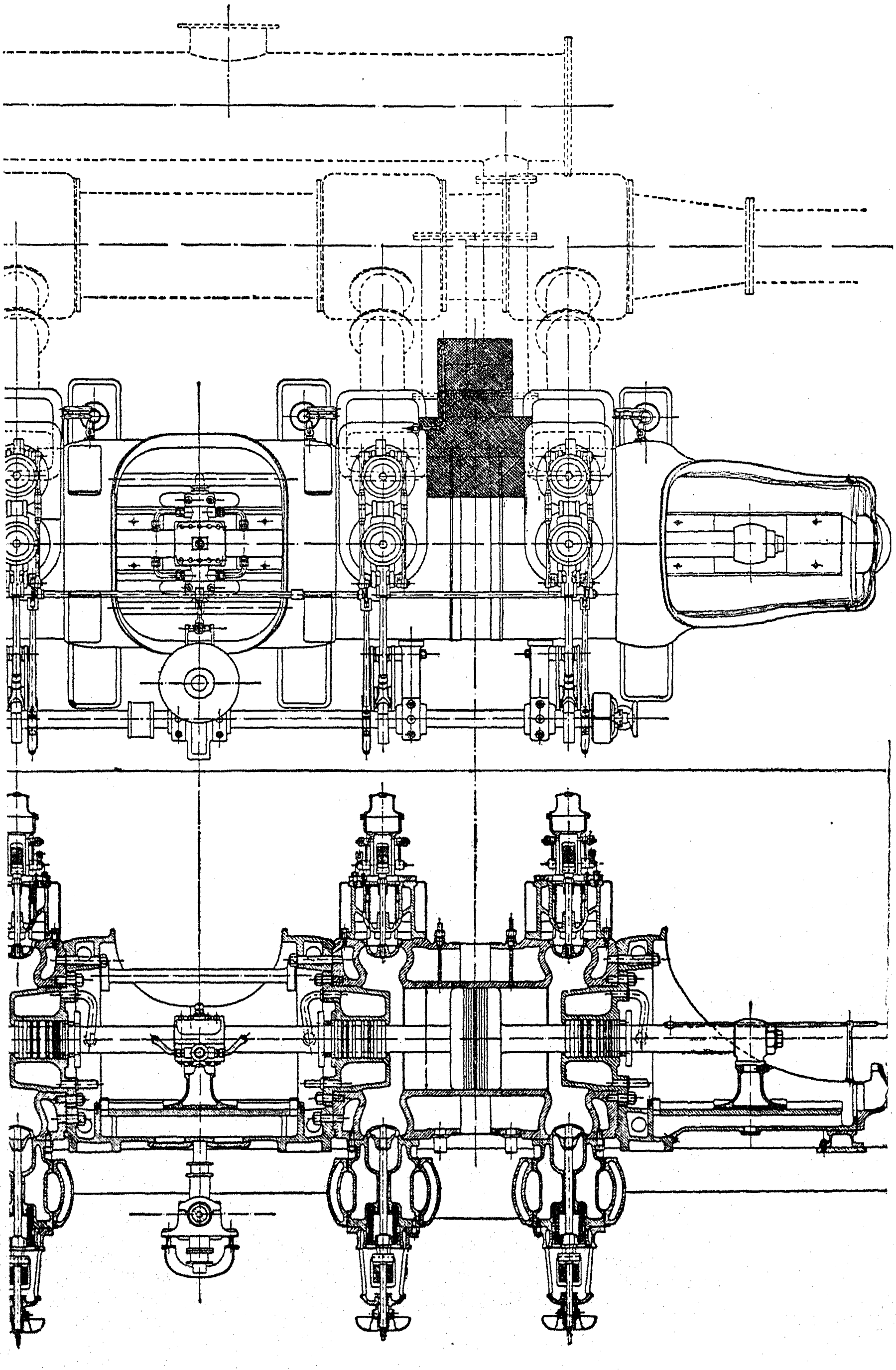


PLATE VI.—TANDEM GAS-ENGINE, BY



THE GUTEHOFFNUNGSHÜTTE, OBERHAUSEN.

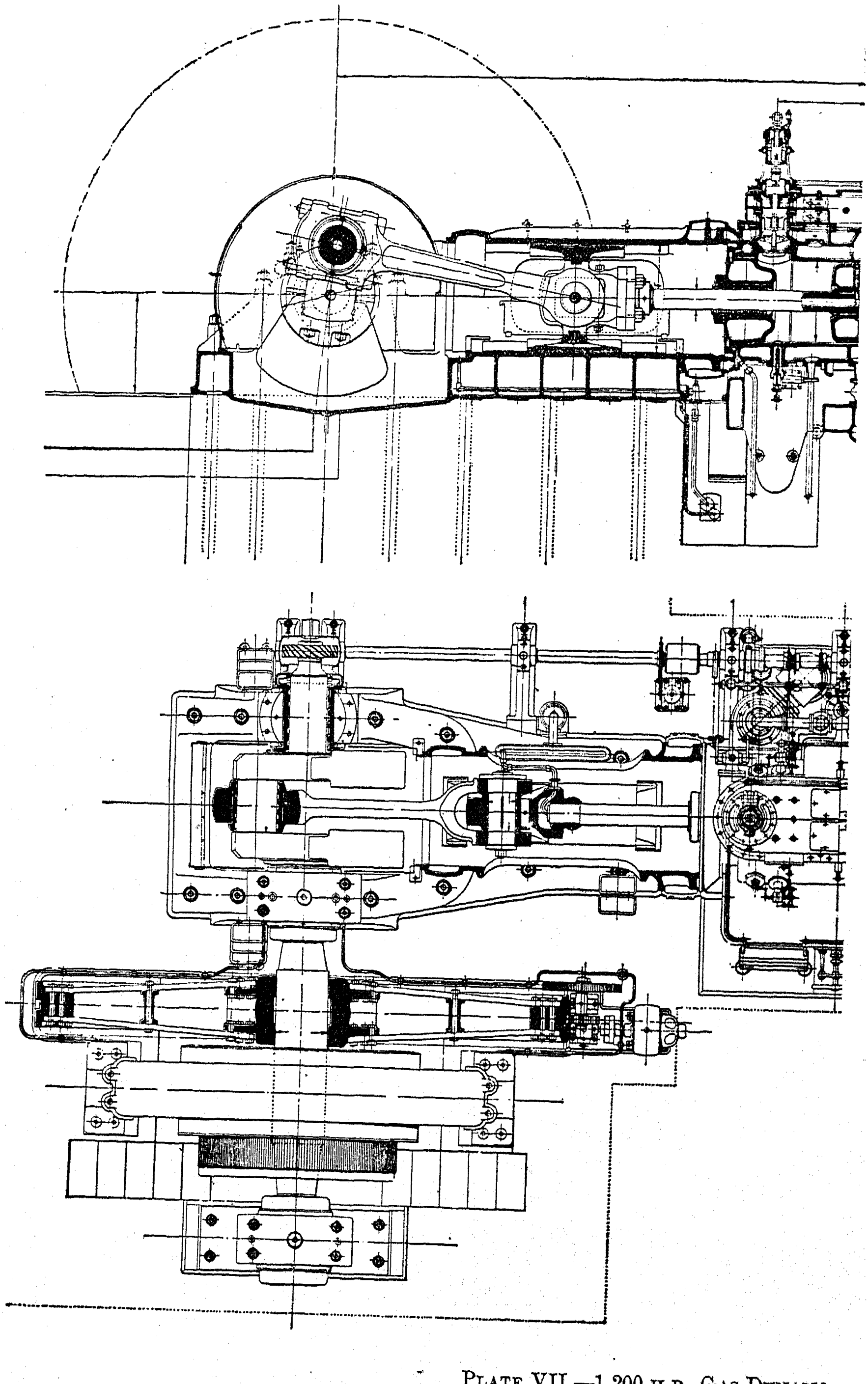
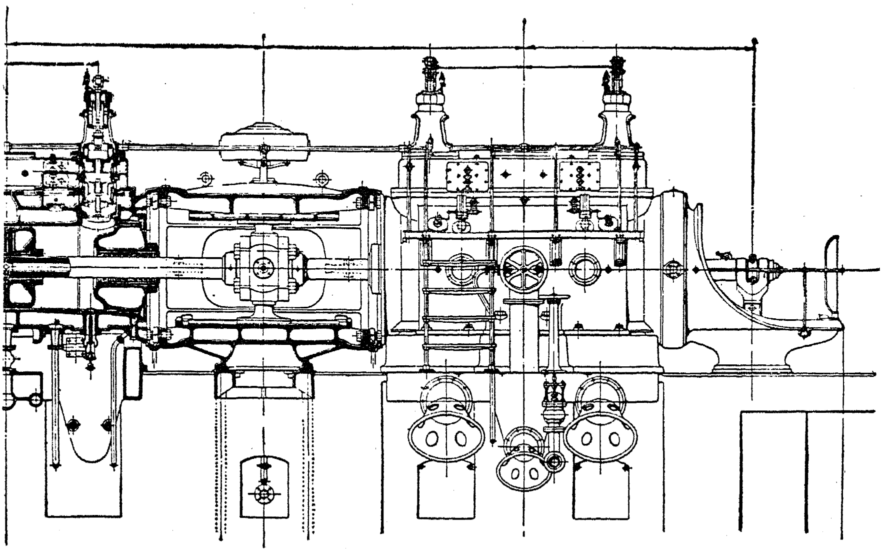
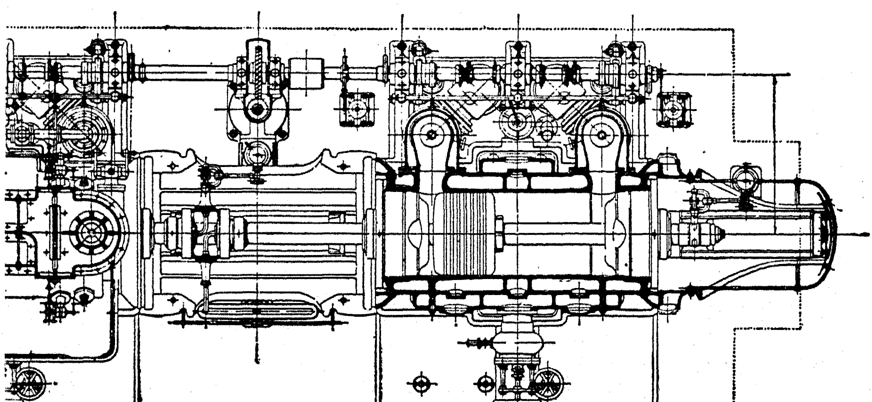


PLATE VII.—1,200-H.P. GAS-DYNAMO,





BY SCHÜCHTERMANN & KREMER, DORTMUND.

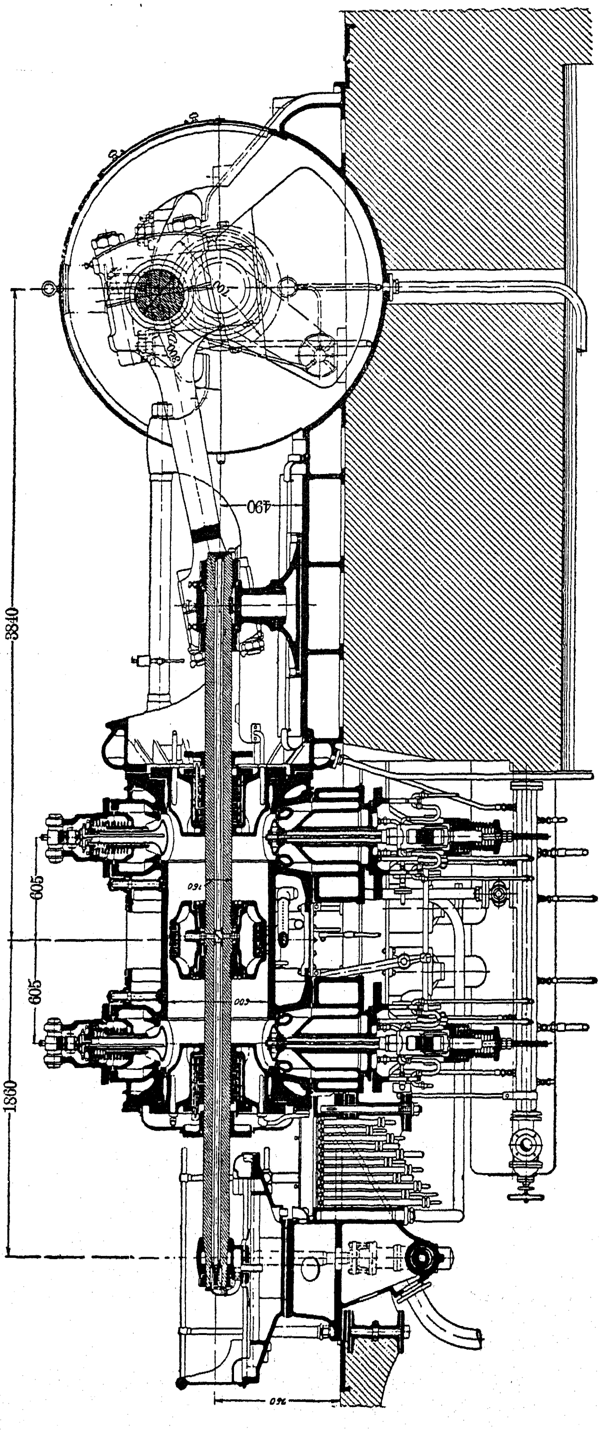


PLATE VIII.—REICHENBACH GAS-ENGINE, BY THE MASCHINENBAU-AKT.-GES. UNION, ESSEN A. D. RUHR. (Longitudinal Section.)

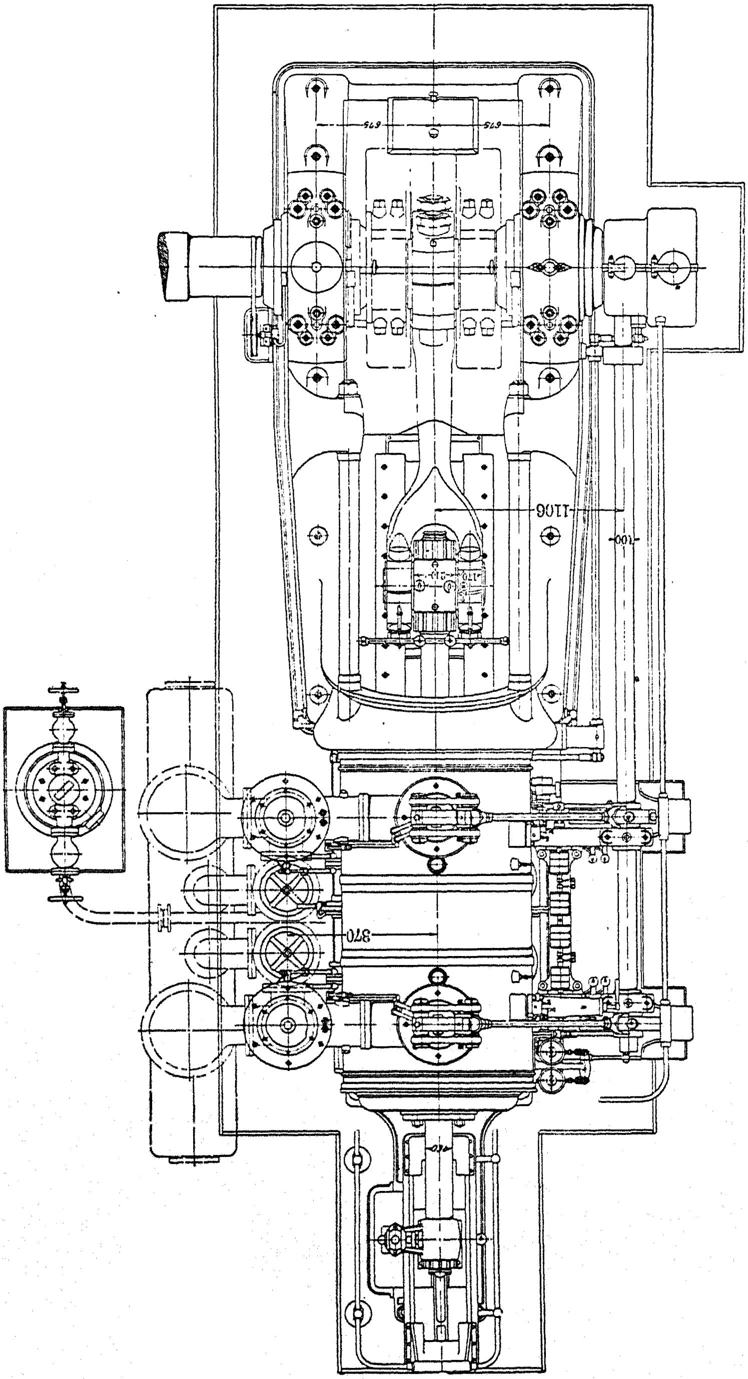


PLATE VIII.—REICHENBACH GAS-ENGINE, BY THE MASCHINENBAU-AKT.-GES. UNION, ESSEN A. D. RUHR. (Plan.)

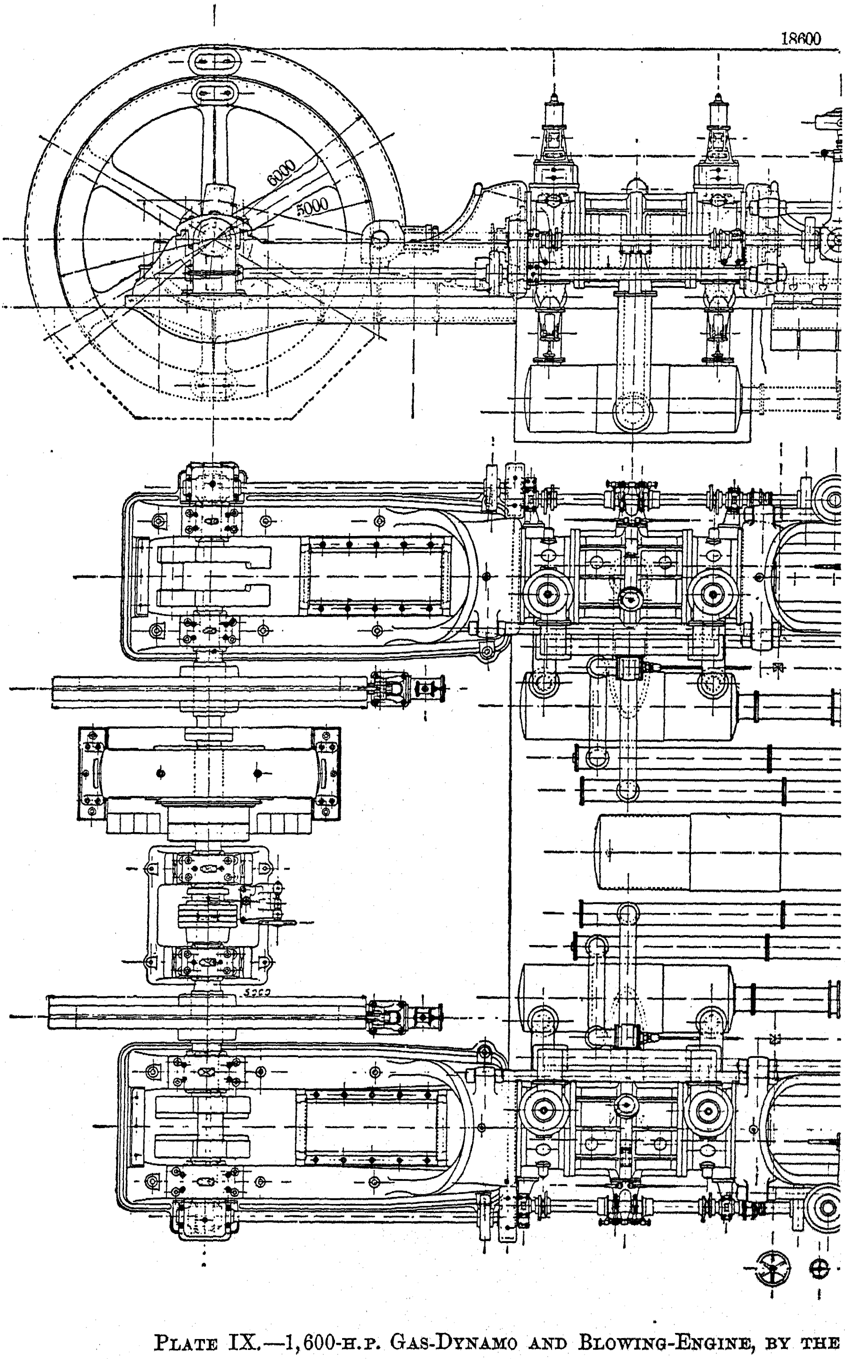
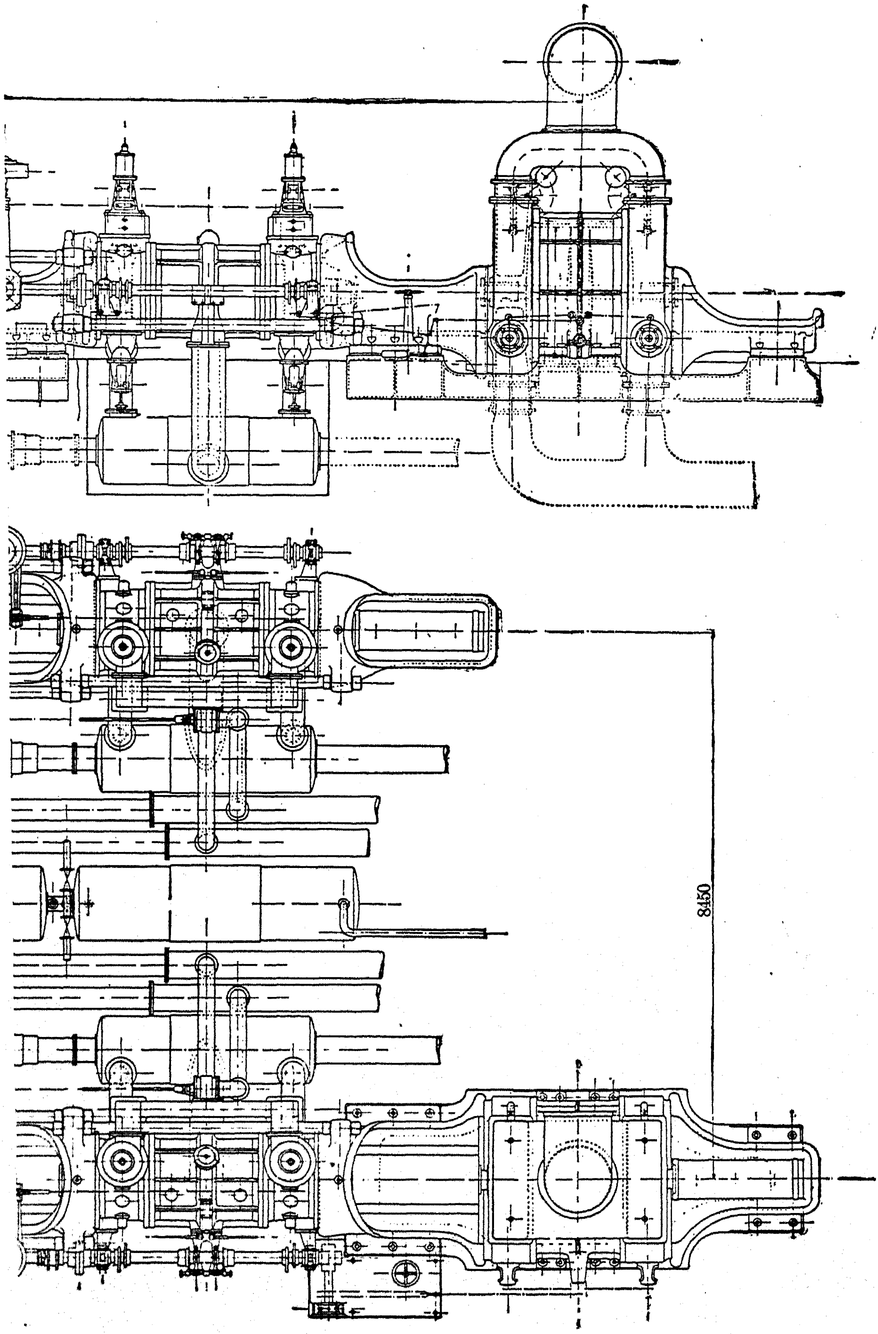


PLATE IX.—1,600-H.P. GAS-DYNAMO AND BLOWING-ENGINE, BY THE



DUISBURGER MASCHINENBAU-AKT.-GES., FORMERLY BECHEM & KEETMAN.

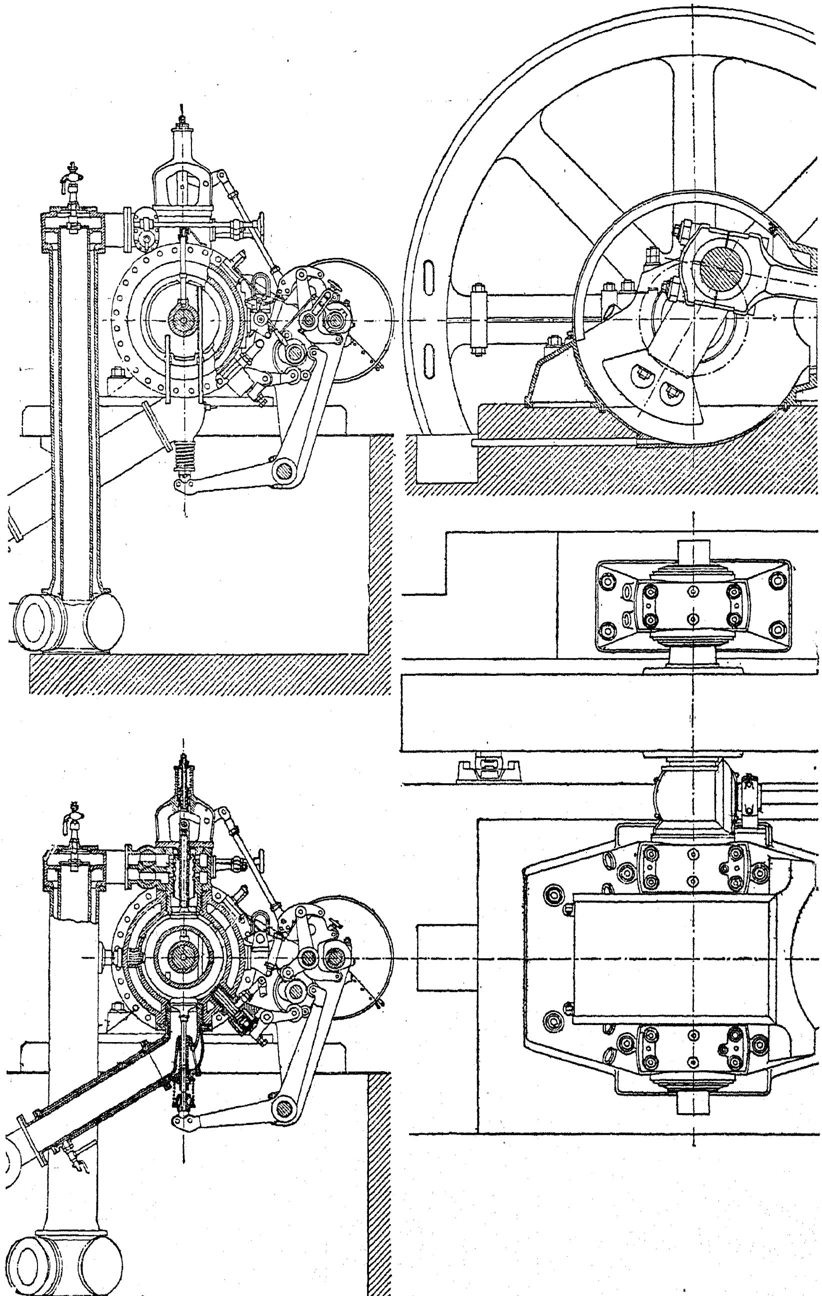
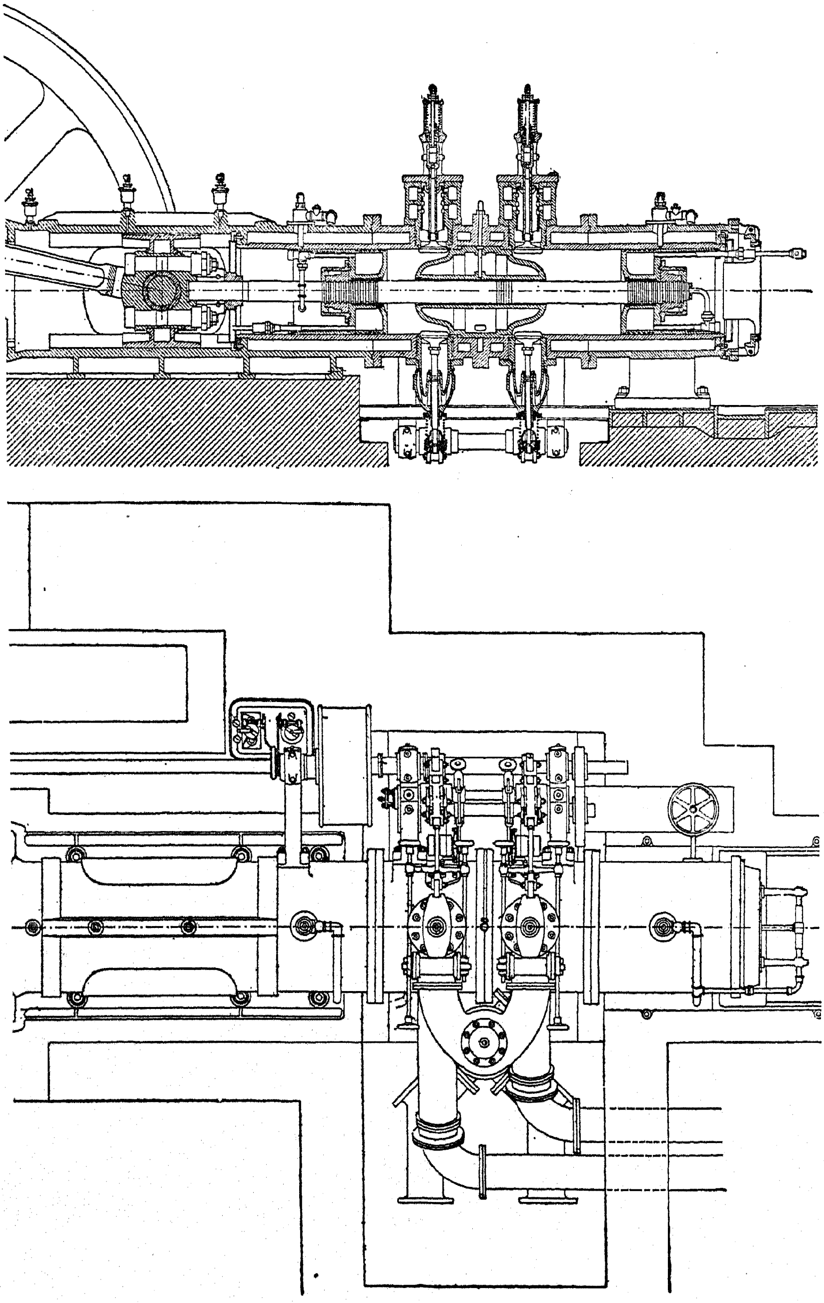


PLATE X.—DOUBLE-ACTING FOUR-CYCLE GAS-ENGINE, BY THE



DINGLER'SCHE MASCHINEN-FABRIK AKT.-GES., ZWEIBRÜCKEN.

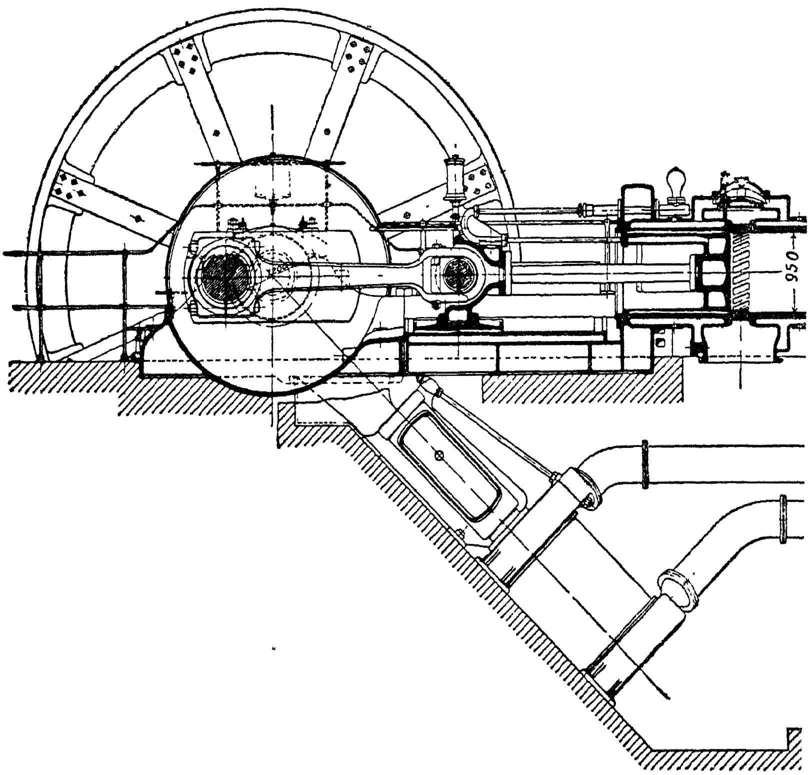
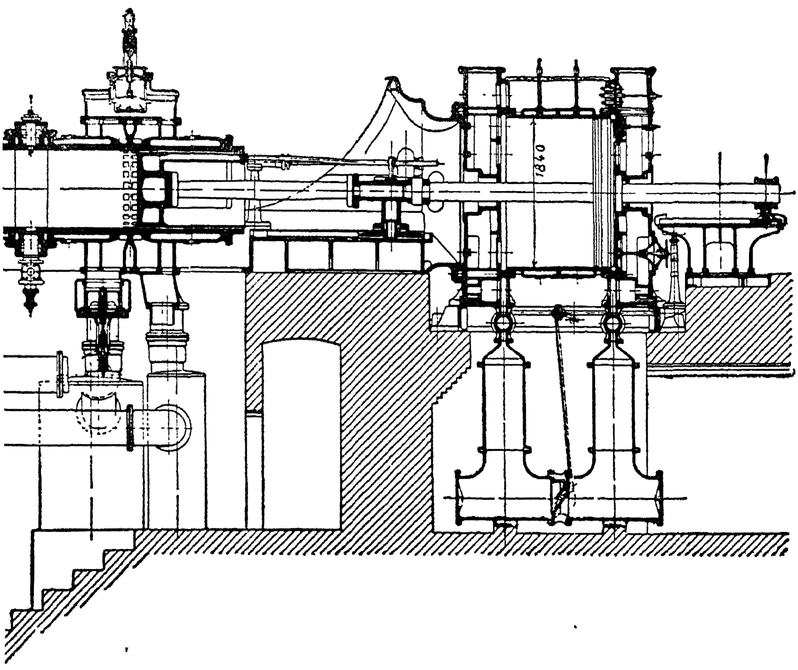


PLATE XI.—TWO-CYCLE ENGINE, OECHELHÄUSER SYSTEM,



BY THE ASCHERSLEBENER MASCHINENBAU-AKT.-GES.

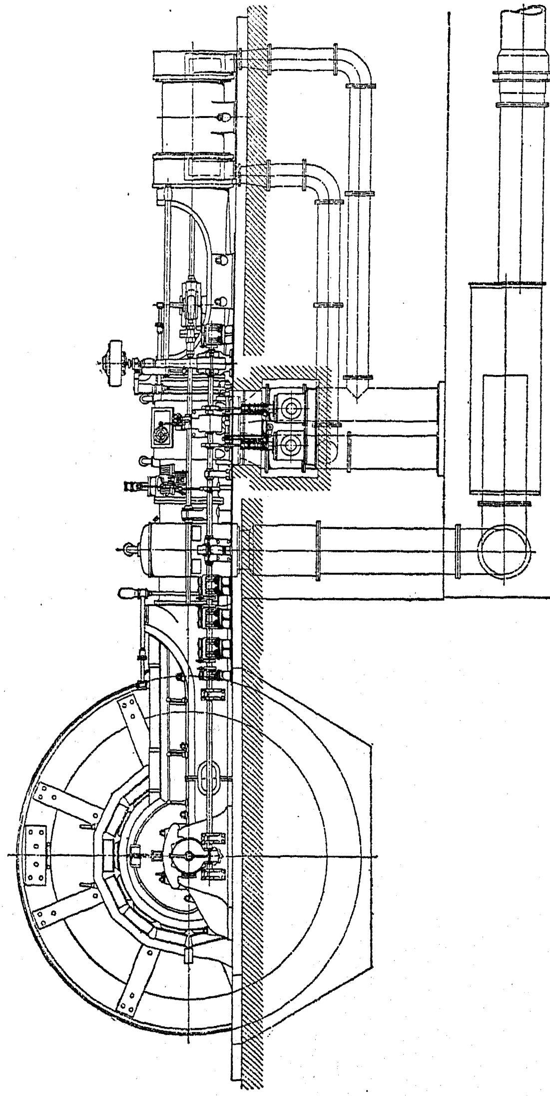


Plate XII.—1,000-h.p. Oechelhäuser Twin Engine, by A. Borsig, Berlin. (Longitudinal Section.)

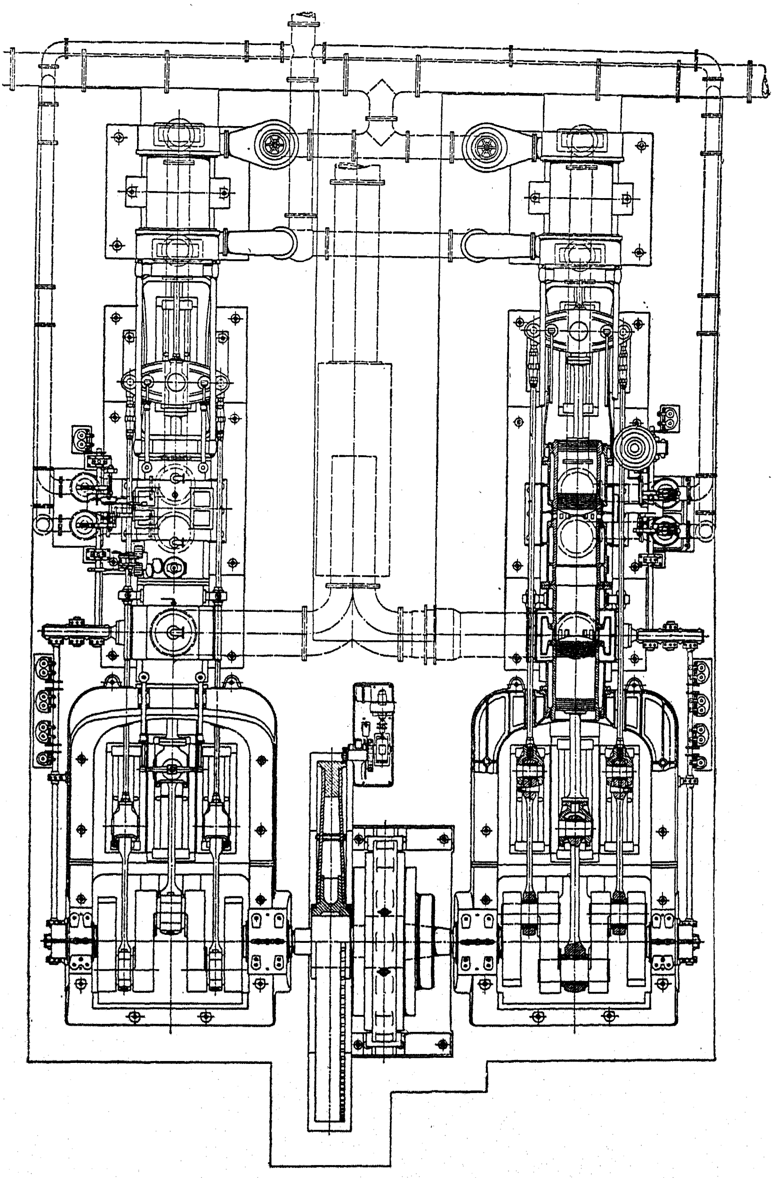


PLATE XII.—1,000-H.P. OECHELHÄUSER TWIN ENGINE, BY A. BORSIG, BERLIN. (Plan.)

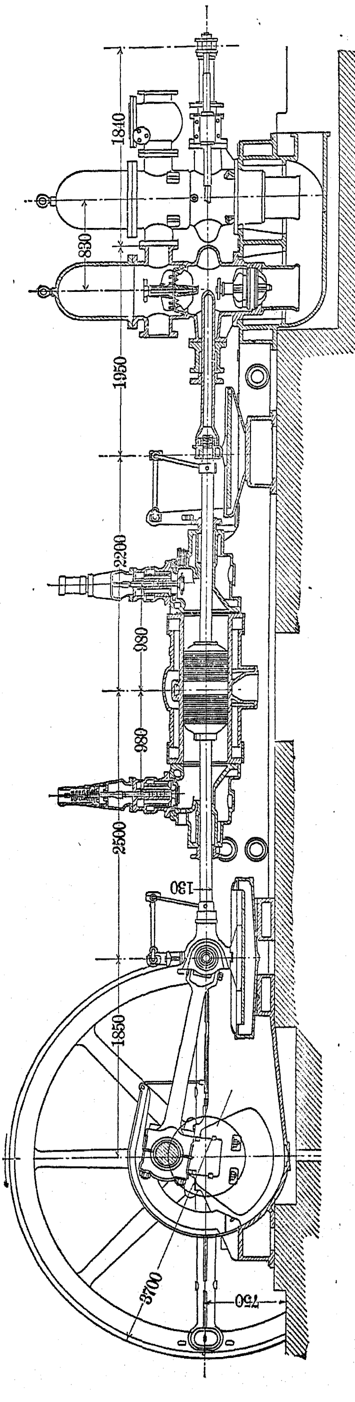


Plate XIII.—Accumulator-Pump with Gas-Driving, by the Siegener Maschinenbau-Akt.-Ges., Siegen. (Longitudinal Section.)

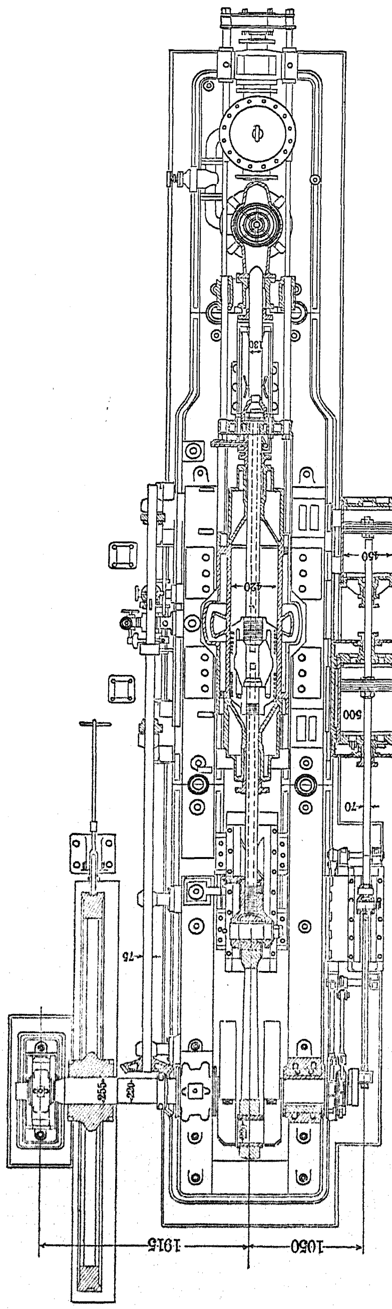


PLATE XIII.—ACCUMULATOR-PUMP WITH GAS-DRIVING, BY THE SIEGENER MASCHINENBAU-AKT.-GES., SIEGEN. (Plan.)

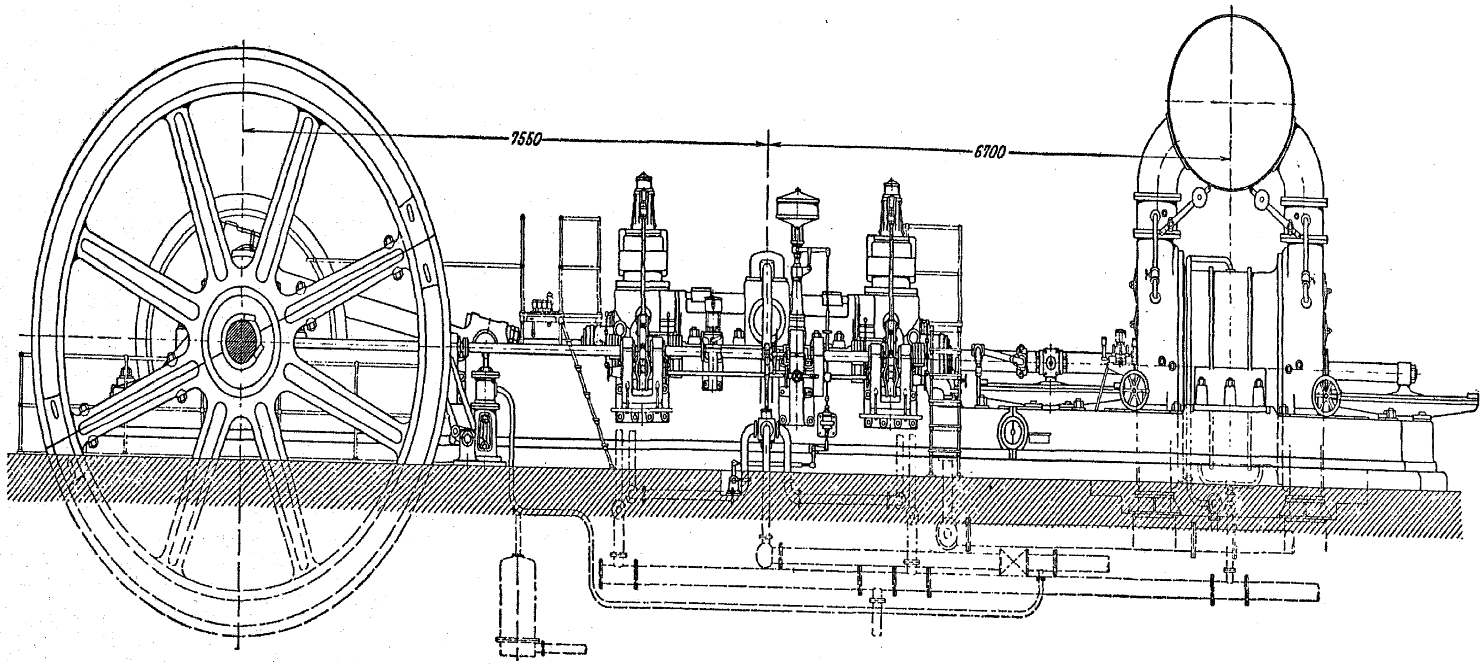


PLATE XIV.—TWIN BLAST-FURNACE GAS-DRIVEN BLOWING-ENGINE, BY THE MASCHINENBAU-AKT.-GES., FORMERLY KLEIN BROTHERS, DAHLBRUCH.
(Longitudinal Section.)

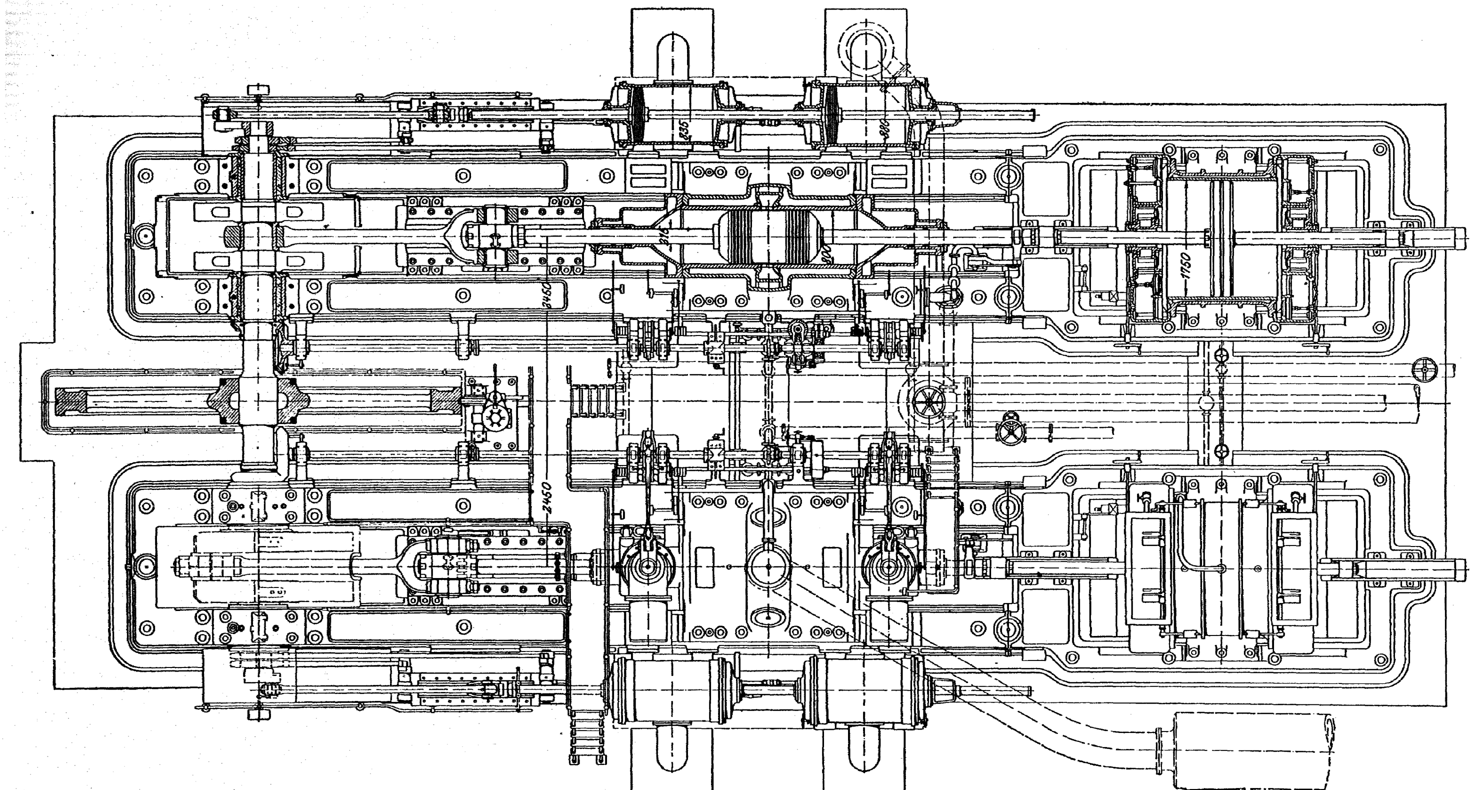


Plate XIV.—Twin Blast-Furnace Gas-Driven Blowing-Engine, by the Maschinenbau-Akt.-Ges., formerly Klein Brothers, Dahlbruch. (Plan.)

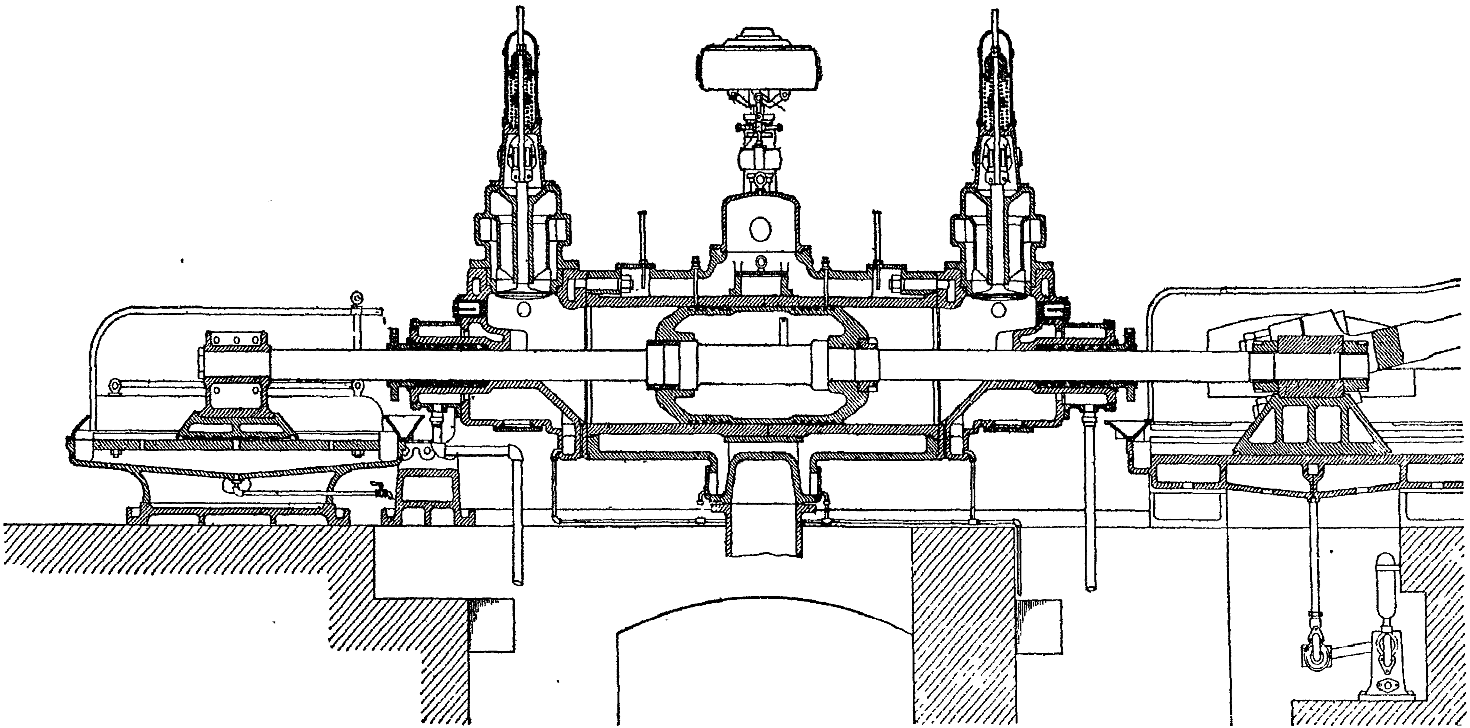


Plate XV.—400-h.p. Körting Gas-Engine, by Körting Brothers, Körtingsdorf. (Longitudinal Section.)

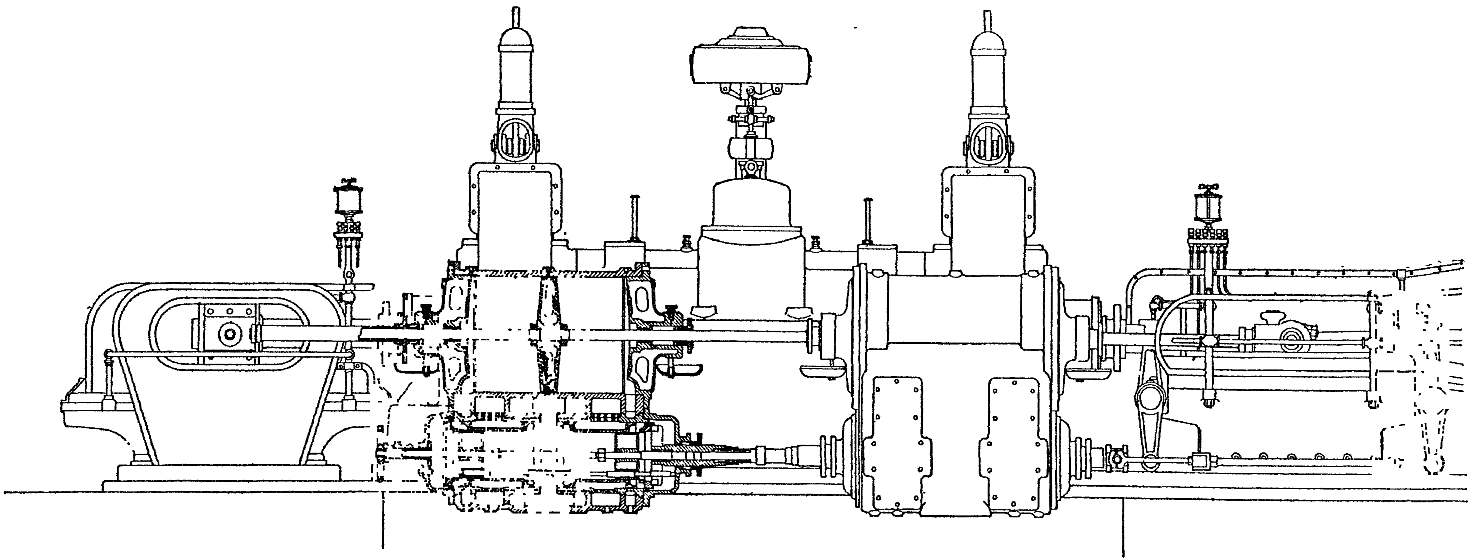


Plate XV.—400-h.p. Körting Gas-Engine, by Körting Brothers, Körtingsdorf. (Longitudinal Section and Elevation.)

Notes on Large Gas-Engines Built in Great Britain, and Upon Gas-Cleaning.*

BY TOM WESTGARTH, MIDDLESBROUGH, ENGLAND.

(London Meeting, July, 1906.)

As papers are placed before you upon large gas-engines in Belgium and Germany, it was considered that some information should be given upon the same subject in Great Britain. I therefore agreed to compile these notes, which I have made very short in view of the somewhat full nature of the other papers presented to this meeting, and considering the complete nature of the many publications which have been made recently upon the subject of large gas-engines.

I give below Tables I. to VI., showing the number and particulars of large gas-engines which have been built or are building by British makers, and in view of the growing size of gas-engines I have concluded that engines of less than 500 h.p. cannot now be included in the category of large gas-engines, and therefore I have not considered engines under that power.

TABLE I.—*"Oechelhäuser" Type Gas-Engines. Built or Building by William Beardmore & Co., Limited, Glasgow.*

No. of Engines.	Indicated Horse-Power Each.	Type.	Nature of Work.	Gas Used.
6	2,500	Twin cylinder.	Dynamo.	Producer.
1	1,850	Single cylinder.	Rolling-mill.	Producer.
4	1,250	Twin cylinder.	Dynamo.	Producer.
2	1,250	Single cylinder.	Dynamo.	Producer.
4	625	Single cylinder.	Dynamo.	Producer.
1	625	Single cylinder.	Rolling-mill.	Producer.
1	625	Single cylinder.	Air compressor.	Producer.
2	500	Single cylinder.	Air compressor.	Producer.
7	500	Single cylinder.	Cement-mill driving.	Producer.

Total, 28 engines; 32,600 indicated horse-power.

* Presented at the Joint Meeting of the Iron and Steel Institute and the American Institute of Mining Engineers, London, July, 1906, and here published under a mutual agreement between the Councils of the two Institutes.

TABLE II.—*"Körting" Type Gas-Engines. Built by Mather & Platt, Limited, Manchester.*

No. of Engines.	Indicated Horse-Power Each.	Type.	Nature of Work.	Gas Used.
1	1,250	Twin cylinder.	Dynamo.	Duff producer.
2	875	Single cylinder.	Dynamo.	Mond producer.
2	625	Single cylinder.	Dynamo.	Duff producer.
1	625	Single cylinder.	Flour-mill.	Mond producer.

Total, 6 engines ; 4,875 indicated horse-power.

TABLE III.—*"Premier" Type Gas-Engines. Built or Building by the Premier Gas-Engine Co., Limited, Nottingham.*

No. of Engines.	Indicated Horse-Power Each.	Type.	Nature of Work.	Gas Used.
2	1,100	Tandem, single-acting.	Dynamo.	Bituminous producer.
1	1,000	Tandem, single-acting.	Blowing.	Blast-furnace.
5	650	Tandem, single-acting.	Dynamo.	Bituminous producer.
4	600	Tandem, single-acting.	Dynamo.	Bituminous producer.
1	600	Tandem, single-acting	Rolling-mill.	Bituminous producer.
11	500	Tandem, single-acting.	Dynamo.	Bituminous producer.
2	500	Tandem, single-acting.	Dynamo.	Coke-oven.
1	500	Tandem, single-acting.	Paper-mill.	Bituminous producer.
1	500	Tandem, single-acting.	Dynamo.	Blast-furnace.

Total, 28 engines ; 16,950 indicated horse-power.

TABLE IV.—*Gas-Engines Built by Willans & Robinson, Limited, Rugby.*

No. of Engines.	Indicated Horse-Power Each.	Type.	Nature of Work.	Gas Used.
2	900	Tandem, double-acting.	Dynamo.	Mason producer.

Total, 2 engines ; 1,800 indicated horse-power.

TABLE V.—*Gas-Engines Built or Building by Crossley Bros., Limited, Manchester.*

No. of Engines.	Indicated Horse-Power Each.	Type.	Nature of Work.	Gas Used.
4	700	Single-acting, *vis-à-vis.*	Alternators.	Producer.
4	700	Single-acting tandem.	Pumping.	Producer.
4	625	Single-acting, *vis-à-vis.*	Pumping.	Coal-gas.
2	610	Four-cylinder single-acting, *vis-à-vis.*	Alternators.	Coal-gas.
5	560	Single-acting, *vis-à-vis.*	Electrolytic work.	Producer.
1	560	Single acting, *vis-à-vis.*	Wire-works.	Producer.
1	560	Single-acting, *vis-à-vis.*	Bleach-works.	Producer.
1	560	Single-acting tandem.	Flour-mill.	Producer.
1	560	Single-acting, *vis-à-vis.*	Blowing.	Blast-furnace.
2	500	Single-acting, *vis-à-vis.*	Electric lighting.	Producer.
2	500	Single-acting tandem.	Electric lighting.	Producer.
1	500	Single-acting, *vis-à-vis.*	Electric lighting.	Coke-oven.
2	500	Single-acting, *vis-à-vis.*	Electrolytic work.	Producer.
1	500	Single-acting, *vis-à-vis.*	Electric tramway.	Producer.
1	500	Single acting, *vis-à-vis.*	Cotton-mill.	Producer.
1	500	Single-acting, *vis-à-vis.*	Mining machinery.	Producer.

Total, 33 engines; 19,360 indicated horse-power.

TABLE VI.—"*Cockerill*" *Type Gas-Engines. Built or Building by Richardsons, Westgarth & Co., Limited, Middlesbrough.*

No. of Engines.	Indicated Horse-Power Each.	Type.	Nature of Work.	Gas Used.
2	1,500	Twin tandem, double-acting.	Tube-rolling mills.	Mond producer.
2	1,500	Twin tandem, double-acting.	Dynamo.	Mond producer.
1	1,000	Tandem, double-acting.	Tube-rolling mill.	Mond producer.
2	950	Tandem, double-acting.	Dynamo.	Blast-furnace.
10	800	Single cylinder, single-acting.	Blowing.	Blast-furnace.
1	800	Tandem, single-acting	Mine fan.	Coke-oven.
2	750	Tandem, double acting.	Dynamo.	Mond producer.
1	650	Tandem, double-acting.	Dynamo.	Mond producer.
1	650	Tandem, double-acting.	Dynamo.	Blast-furnace.

Total, 22 engines; 20,500 indicated horse-power.

It will be observed no mention is made in Tables I. to VI. of engines built by Messrs. Hornsbys of Grantham, Rodger & Co. of Glasgow, Campbell of Halifax, Fielding & Platt of Gloucester, the National Company of Ashton-under-Lyne,

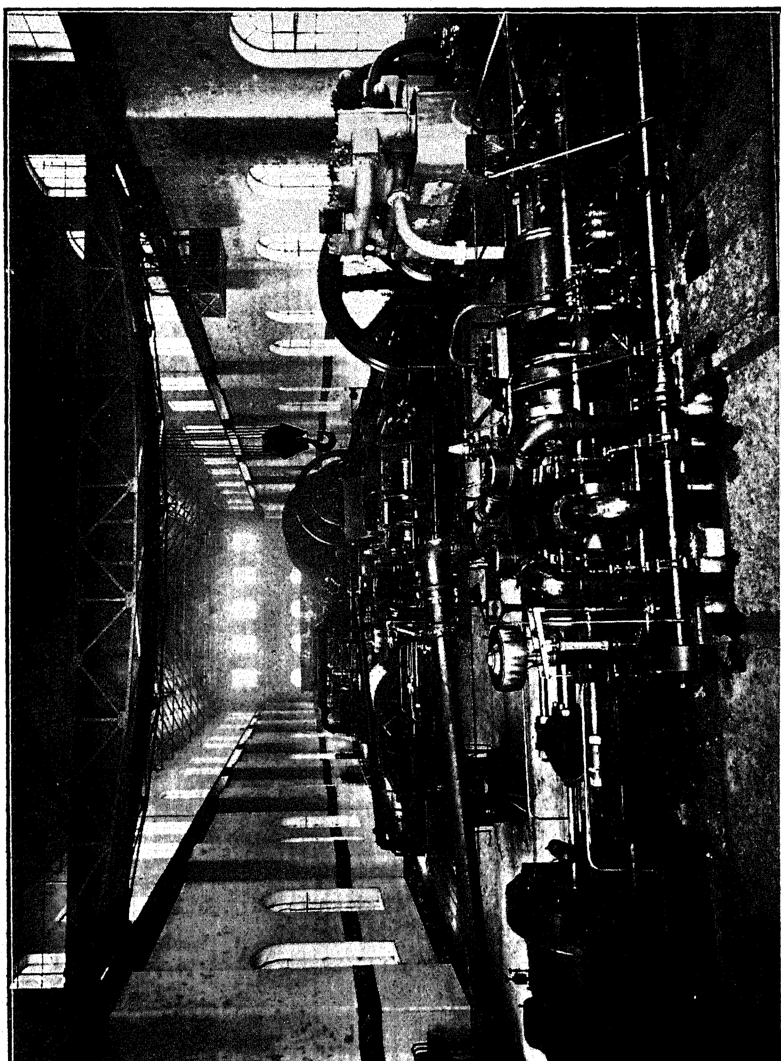


Fig. 1.—Oechelhäuser Engines Built by William Beardmore & Co., Ltd.; 6,625 i.h.p. in Seven Units.

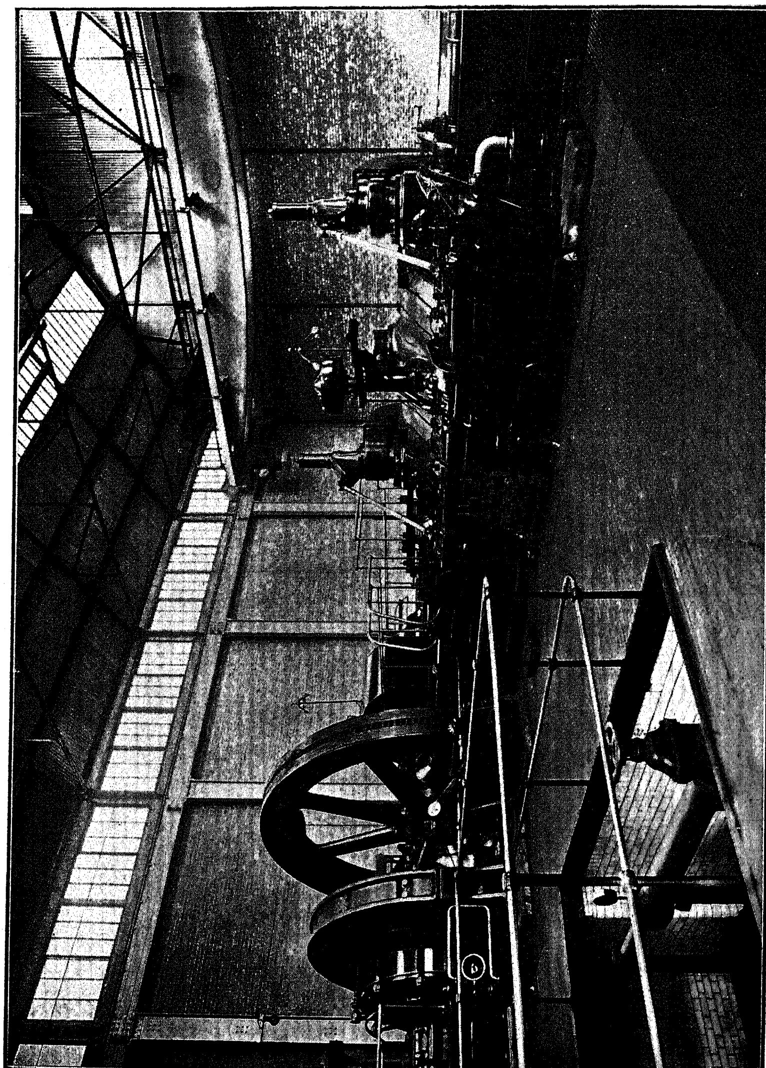


Fig. 2.—Two 875-I.H.P. Körting Engines Built by Mather & Platt, Ltd.

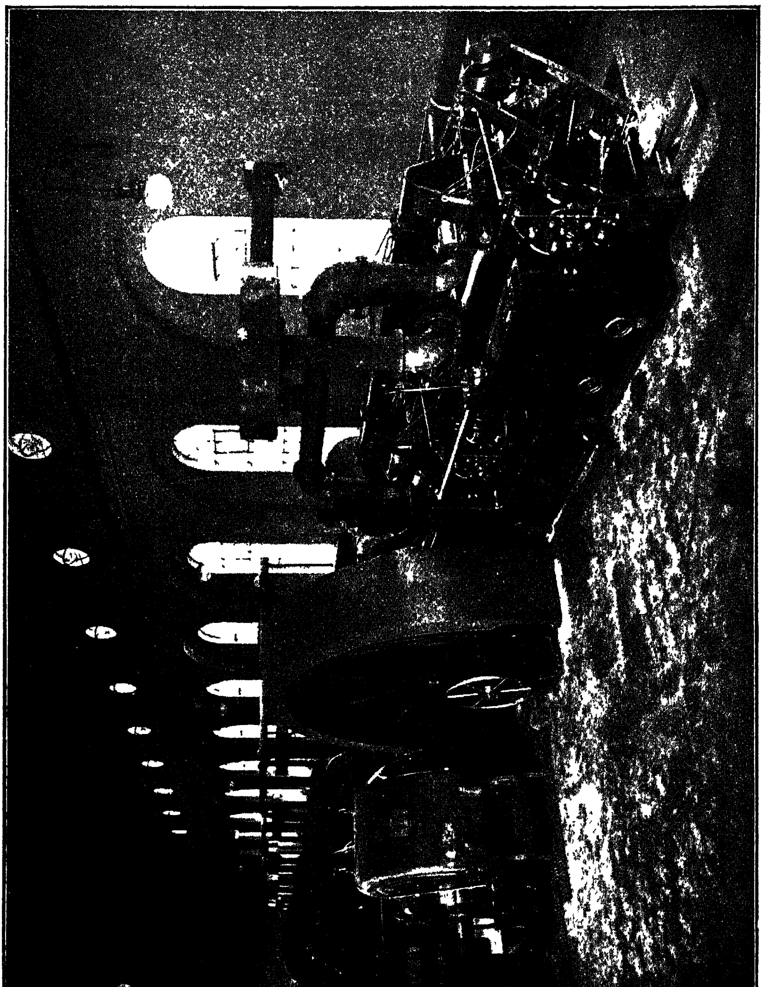


FIG. 3.—650-H.P. ENGINE BUILT BY THE PREMIER GAS-ENGINE CO., LTD.

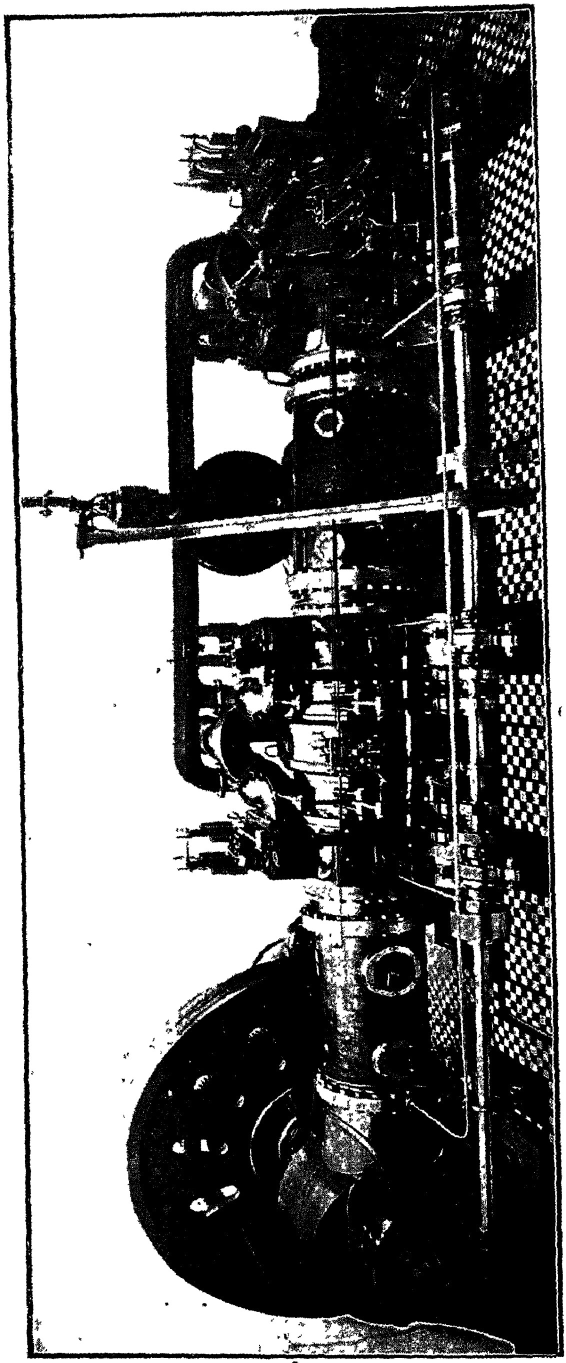


Fig. 4.—900-h.p. Engine Built by Willans & Robinson, Ltd.

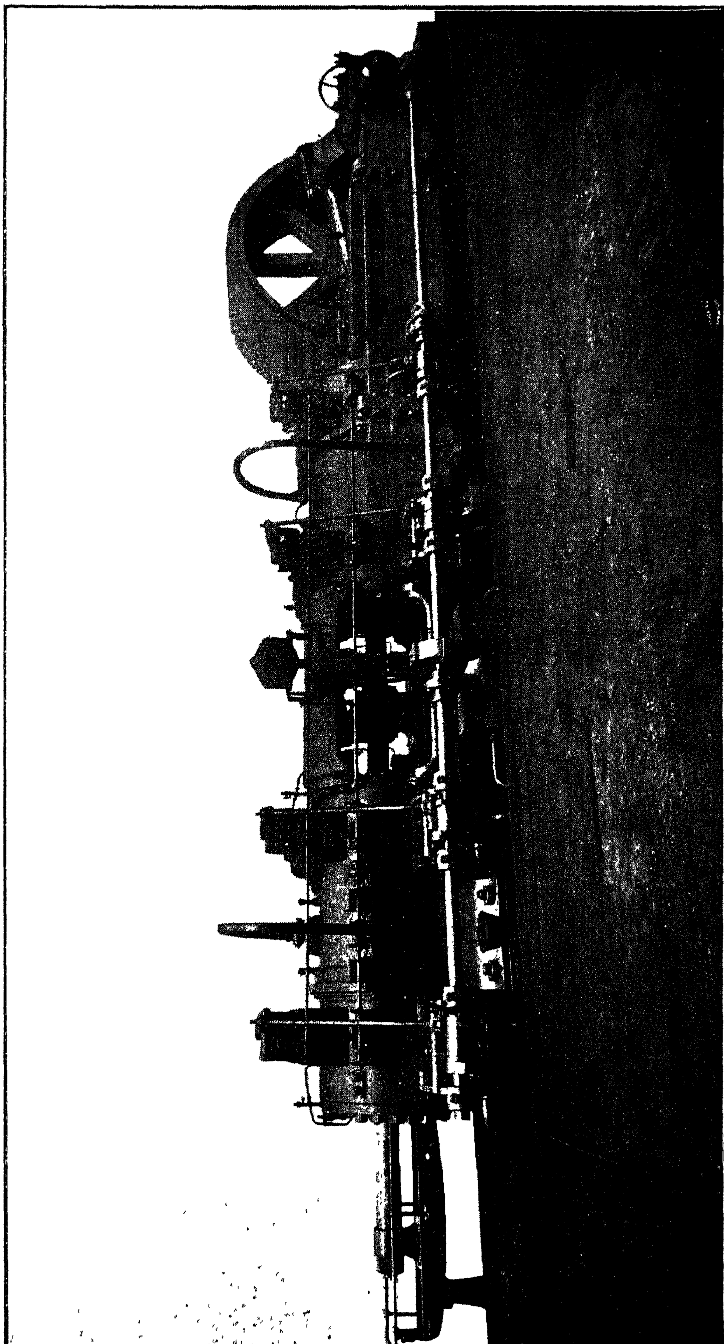


Fig. 5.—625-i.h.p. Engine Built by Crossley Bros., Ltd.

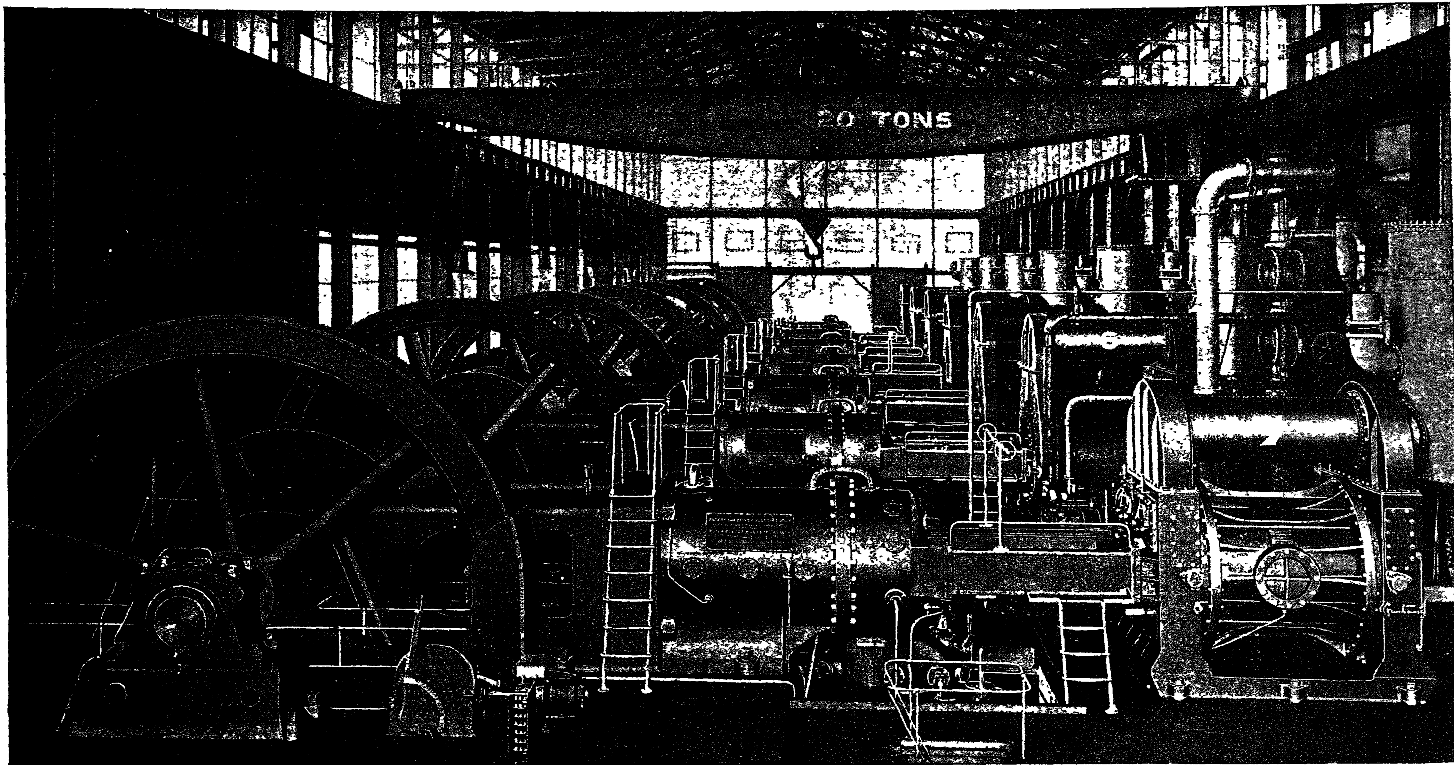


Fig. 6.—Gas-Driven Blowing-Engines Built by Richardsons, Westgarth & Co., Ltd.

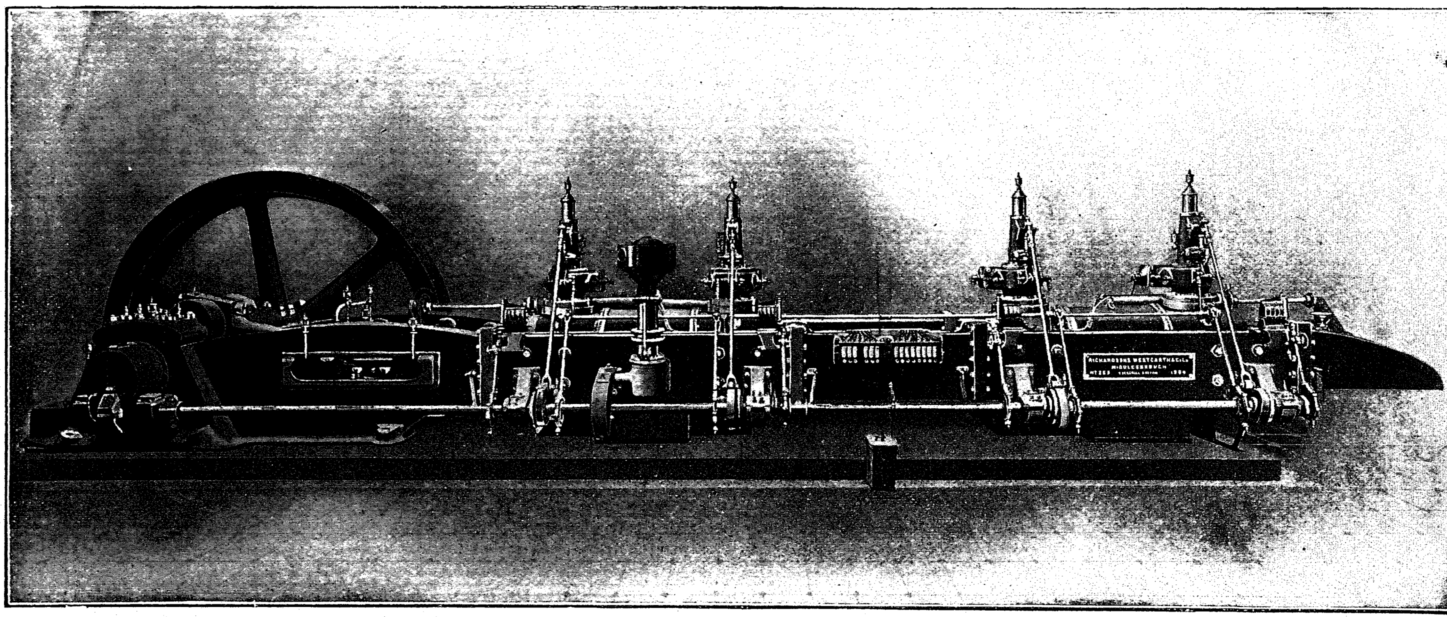


Fig. 7.—Tandem Double-Acting Gas-Engine Built by Richardsons, Westgarth & Co., Ltd.

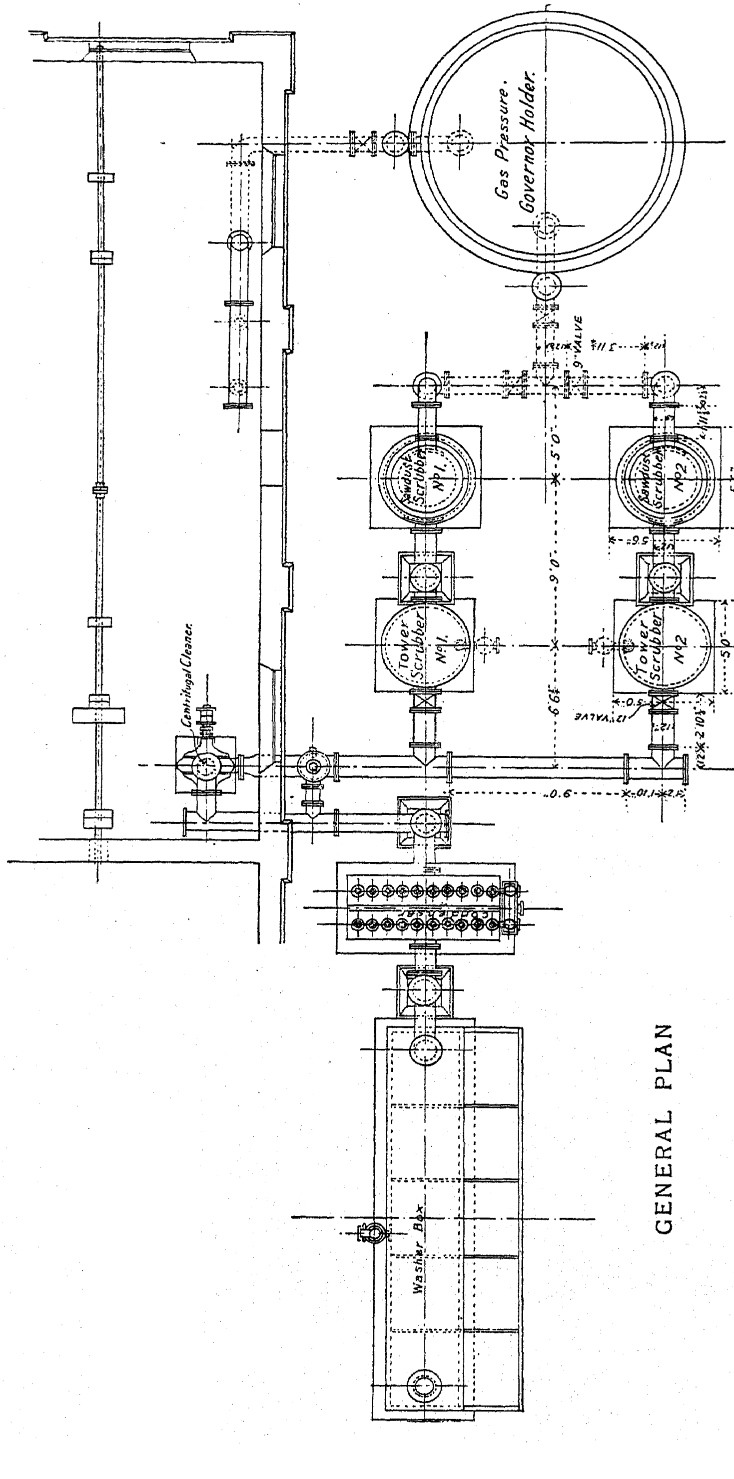


FIG. 8.—THWAITE BLAST-FURNACE GAS-CLEANING PLANT INSTALLED IN 1889 AT THE WORKS OF THE SHEEPBRIDGE IRON CO.

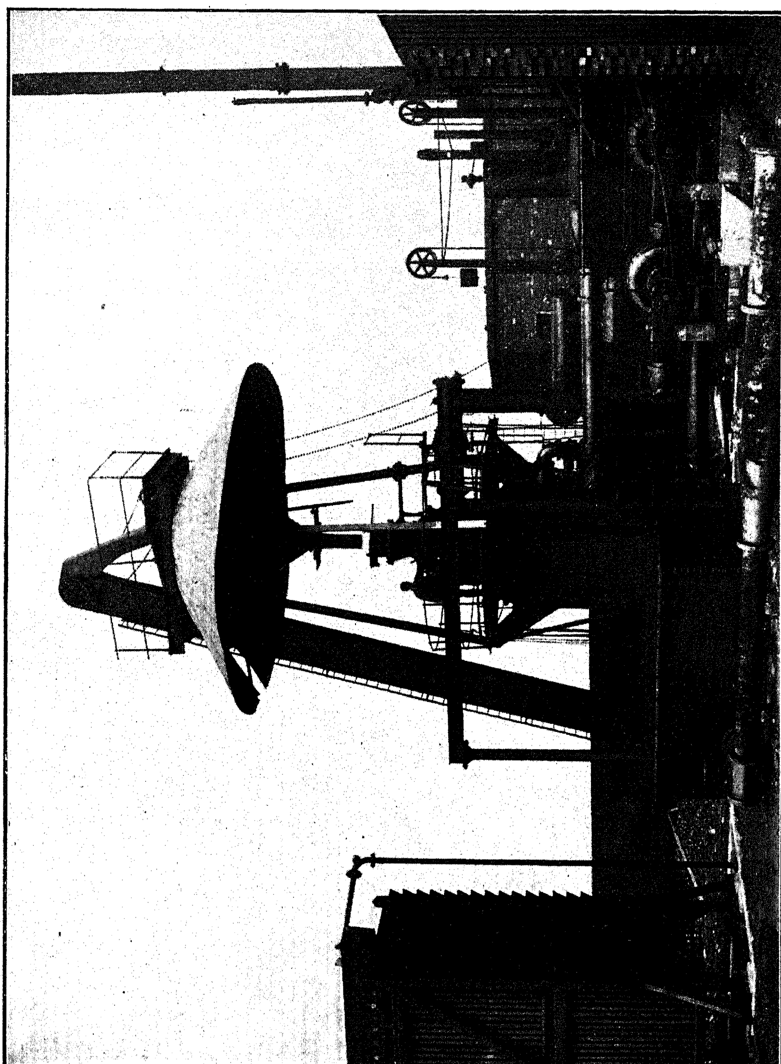


FIG. 9.—GAS-CLEANING APPARATUS FOR PRODUCER-GAS BUILT BY THE POWER GAS CORPORATION.

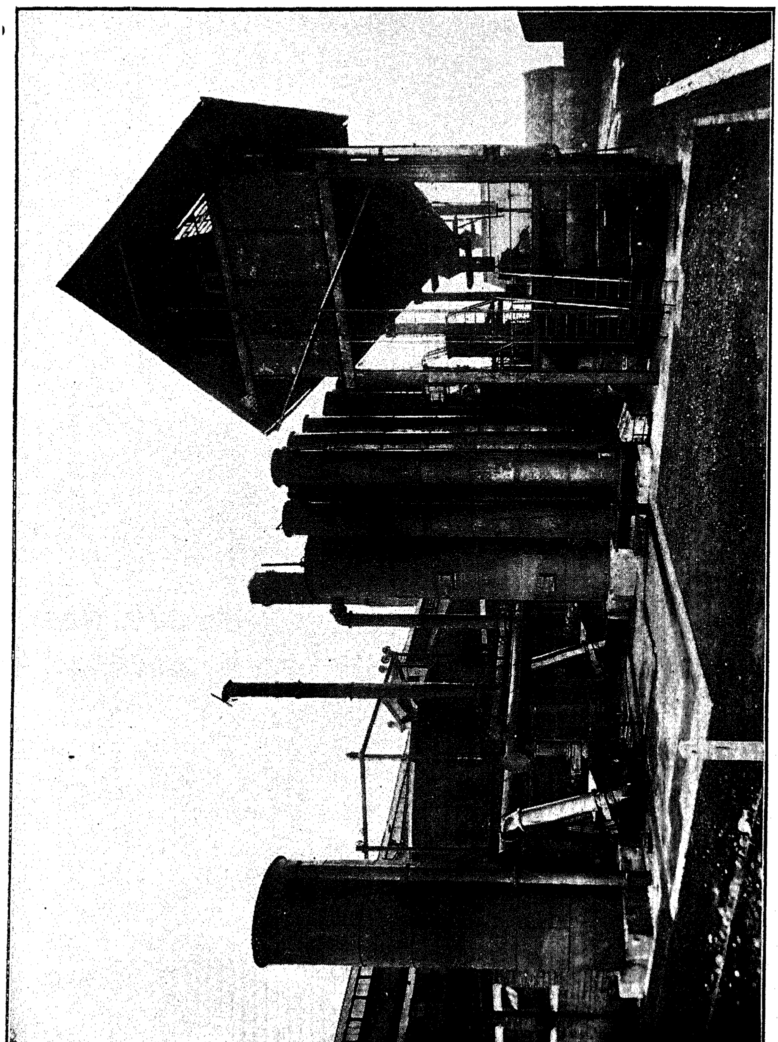


FIG. 10.—GAS-CLEANING APPARATUS FOR PRODUCER-GAS BUILT BY MASON'S GAS POWER CO., LTD.

Tangyes of Birmingham, or the Westinghouse Company of Manchester, which is accounted for by the fact that none of these firms have as yet built engines of 500 h.p. or upwards, although they are all building engines of fair size.

It will be noticed that all the British builders of large gas-engines are using the "four-cycle" system, except the builders of Körting and Oechelhäuser engines, who work upon the "two-cycle" system. It will also be noticed that large gas-engines are gradually coming into use in Great Britain for general purposes; *i.e.*, in addition to blowing and dynamo work, they are being applied to rolling-mills and for general manufacturing purposes, cotton-mills, cement-works, etc.

To illustrate the extent to which gas-engines of large size are now being used in Great Britain, I have included in the paper illustrations of some of the principal installations in the country.

Fig. 1 shows the installation of Oechelhäuser engines built by Messrs. Beardmore for their new shipyard at Dalmuir. This installation is of 6,625 i.h.p. in seven units.

Fig. 2 shows part of an installation of Körting engines by Messrs. Mather & Platt, consisting of two 875-i.h.p. engines now running at a chemical works.

Fig. 3. Engines supplied to Messrs. Bayliss, Jones & Bayliss by the Premier Co., the engine in the foreground being of 650 h.p.; the other engines are of less than 500 h.p., and therefore do not come within the scope of this paper.

Fig. 4. One of a pair of 900-i.h.p. engines built by Messrs Willans & Robinson, and running at Reading.

Fig. 5 is an illustration of one of Messrs. Crossley Brothers' latest engines, of 625 i.h.p.

Fig. 6 shows a large installation of gas-driven blowing-engines built by Messrs. Richardsons, Westgarth & Co., Limited, of Middlesbrough, for the Cargo Fleet Iron Co., Limited. There are seven engines, of a total of 5,600 i.h.p.

Fig. 7 shows Messrs. Richardsons, Westgarth & Co.'s latest type of tandem double-acting gas-engines for dynamo and general work, and is shown as illustrating the latest development in this type of engine; the arrangement being one which, with alteration in detail, has been adopted generally by all the principal makers of gas-engines working on the "Otto" cycle.

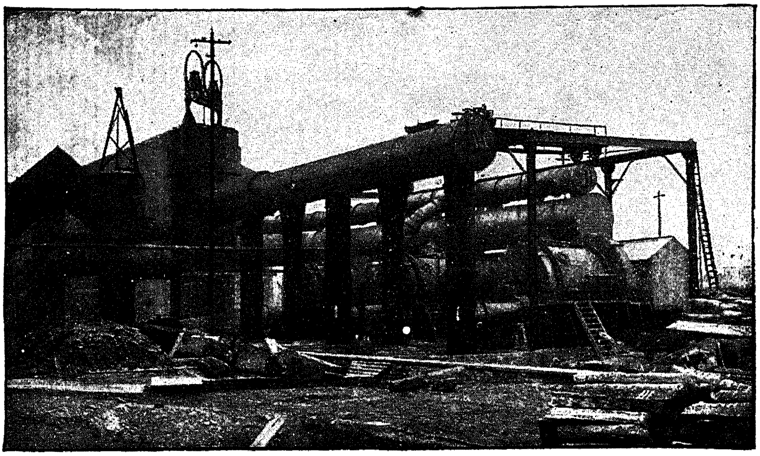


FIG. 11.—THEISEN'S GAS-CLEANING APPARATUS AT THE WORKS OF THE CARGO FLEET IRON COMPANY.

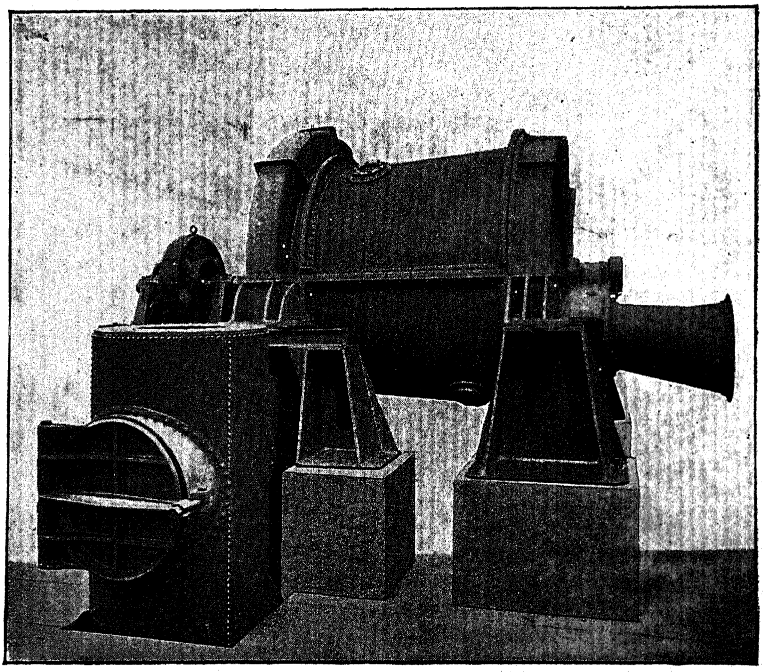


FIG. 11A.—THEISEN'S GAS-CLEANING APPARATUS.

It is thought that any notes upon large gas-engines in Great Britain should be accompanied by a reference to gas-cleaning, this being such an important matter in connection with the proper running of the engines.

Most engineers commenced cleaning the gas for engines by passing it through ordinary fans with water. It was soon found that two or more fans in series would be necessary, and that a considerable quantity of water and much power were required, the result not always being satisfactory, particularly if the gas had not previously been very well cleaned and cooled by passing through large dust-catchers and long mains. A difficulty also occurs with the moisture absorbed by the gas in the fans, and it has been found necessary to supplement the fans by various cleaning and drying devices.

One of the pioneers in dealing with the cleaning of gas for engines was Mr. B. H. Thwaite, and Fig. 8 gives particulars of his apparatus, which is cleaning blast-furnace gas at Sheepbridge. A similar plant is also working at Ardsley.

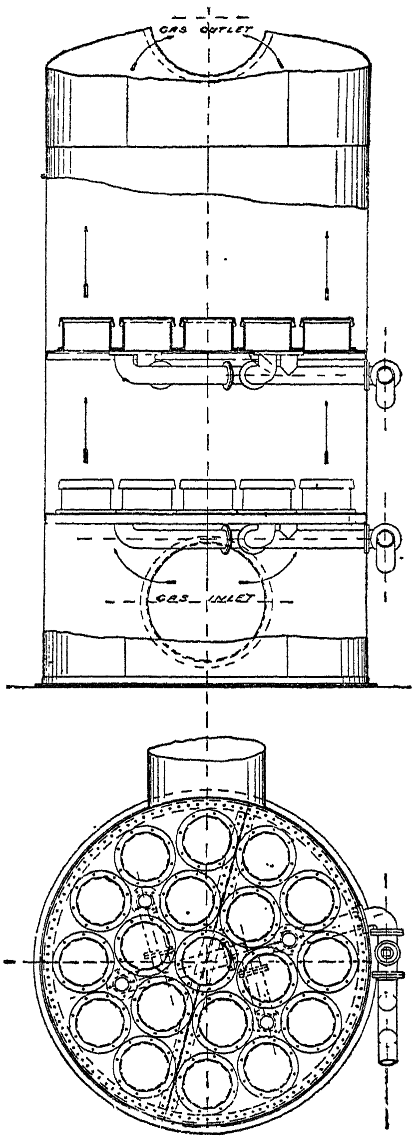


FIG. 12.—TAR-EXTRACTION APPARATUS OF THE SUMMERLEE COMPANY.

Other methods of cleaning the gas upon much the same lines have been worked out in connection with producer-gas by the Power Gas Corporation of Westminster, and Mason's Gas Power Co., Limited, of Manchester. Fig. 9 shows a plant

installed by the Power Gas Corporation at West Gorton, and Fig. 10 a plant erected by Mason's Company at Reading. It will be observed that in both cases fans are used in conjunction with cooling-towers, etc.

Figs. 11 and 11A represent a large installation of Theisen's patent gas-cleaning apparatus, erected at the works of the Cargo Fleet Iron Company by Richardsons, Westgarth & Co., Limited, of Middlesbrough, which shows the latest practice in cleaning blast-furnace gas. This apparatus is able to clean ordinary blast-furnace gas down to 0.002 g. per cu. m. with the use of less than 1 liter of water per cu. m. of gas. It will be observed that a vapor-separating apparatus is fitted to the outlet of the Theisen cleaner, which has the effect of thoroughly drying the gas.

Fig. 12 illustrates the Summerlee Company's patent apparatus for extracting the tar from gas after it has passed through a by-product recovery plant.

I have not entered into any detailed description of the various gas-cleaning apparatus named above, my object in these notes being merely to call attention to the various systems for the benefit of those who require general information. Nor have I made any mention of Zschocke, Bian, or Sahlin gas-cleaning plants, because, although these are receiving some attention in this country, they have not been used here so far as I know.

The Crystallography of Iron.*

BY F. OSMOND AND G. CARTAUD, PARIS, FRANCE.

(London Meeting, July, 1906.)

I. INTRODUCTION.

WE have already devoted two previous memoirs to this question. In the first[1] we collated and discussed the existing literature on the subject; in the second,[2] we described the crystalline forms obtained by the reduction, at different temperatures, of ferrous chloride by hydrogen or by zinc-vapor. The conclusion from these researches was that the three allotropic states of iron—α, stable below $A2$; β, stable between $A2$ and $A3$; and γ, stable above $A3$—all crystallize in the cubic system. The differences observed were such as are customarily encountered in the crystallographic records of many minerals of which allotropic varieties or isomerides are not known, and did not conform to the ordinary definition of polymorphism.

It is, however, improbable that allotropic transformations, which are placed beyond doubt by a series of positive facts, do not involve some changes in the intimate structure.

As a matter of fact, the crystals we did obtain were too small to allow of a precise examination, and this might introduce an element of uncertainty in the firm establishment of our conclusions. Professor H. Le Chatelier was even led to think from some of our results that γ-iron could very well be a rhombohedron simulating a cube, and not a true cube, and it will be remembered that the crystalline form of bismuth was for a long time incorrectly regarded as cubic.

Allotropy does not, however, necessarily involve such a deduction. Without departing from the cubic system at all, a whole

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* Presented at the Joint Meeting of the Iron and Steel Institute and the American Institute of Mining Engineers at London, July, 1906, and here published under a mutual agreement between the Councils of the two Institutes.

[1] *Annales des Mines*, vol. xvii., pp. 110 to 165 (1900).

[2] *Ibid.*, vol. xviii., pp. 113 to 153 (1900).

series of variants may be found incompatible among themselves, and yet compatible with the same external forms. But as a study of the latter would scarcely lead to conclusive results concerning the structure of iron, other methods had to be sought to solve the question.

Optical methods are not available, but the morphological and other characters capable of yielding useful information are as follows:

1. Deformation figures:
   Continuous (*a*).
   Discontinuous—effaceable: lines of translation and folding (*b*); not effaceable: mechanical twinning (*c*).
2. Congenital twinning.
3. Twinning resulting from annealing after deformation.
4. Mechanical properties functional of the crystalline orientation.
5. Corrosion figures.
6. Synchronous crystallization figures.
7. Segregation figures.

Before we describe our experiments and the results of our observations, we shall indicate the principles of these different methods, some of which are but slightly known.

1. *Deformation Figures.*

In a paper published in collaboration with Mr. Frémont[3] we have expressed the opinion that a crystalline body may in a sense be considered both as cellular and amorphous; the former in so far as it is formed of polyhedral grains, each of which is a crystalline element of definite orientation, and the latter when the deformations are governed only by the direction of the strains and are independent of the crystalline structure.

Whence arise three kinds of deformation, which may be called crystalline, cellular, and banal. In the present work the question of cellular deformation will not arise, as our operations are restricted to isolated crystals. We are chiefly engaged in crystalline deformations. With regard to banal deformations, we shall not study them for their own sake, but only as distinctive characters when we encounter them in course of work.

[3] *Revue de Métallurgie,* vol. i., pp. 11 to 45 (1904).

Every deformation has some sort of general configuration, which may be called its silhouette, and which is the boundary of elementary deformations. It is the area of a previously polished body that loses its polish when the deformation takes place. A silhouette is naturally continuous and closed. Its form on an ordinary metal with cellular structure depends only on the form of the sample and on the nature or direction of the strains; but it ceases to be the same on an isolated crystal, for then its form can be in relation with the crystalline symmetry.

1 (*a*). *Continuous Crystalline Deformations; Silhouettes.*—If a sharp conical point is applied to a flat surface of a plastic crystal previously polished, from which all skin-hardening has been removed, a permanent deformation is obtained around the cone of penetration, the silhouette of which is not circular, as it would be on an amorphous or finely grained bed, but presents a definite form characteristic of, firstly, the crystallographic system to which the crystal belongs, and secondly, of the crystallographic orientation of the surface concerned.

These silhouettes can therefore give at least two kinds of information; that is, they can indicate the system of crystallization to which a crystalline body belongs, if that knowledge is required, or if the system is already known they can approximately orient a slice of unknown orientation.

1 (*b and c*). *Discontinuous Deformations.*—Total deformation inclosed in the perimeter of a silhouette is an assemblage of elementary deformations which, in part at least, are discontinuous and in lines. The lines may be straight or curved. In crystalline bodies the straight lines generally have a definite crystallographical orientation. These lines, when obliterated by polishing, may or may not reappear after the re-polished surface is etched by a suitable reagent.

It is generally admitted that a crystalline body is an aggregate of identical molecular polyhedra of the same orientation, and having their centers of gravity at the intersections of a reticulated complex.[4]

Now, it is an acknowledged fact that in a given crystalline body certain definite reticulate planes T (Fig. 1, in perspective), called planes of translation, are susceptible to displacement

[4] De Lapparent, *Cours de Minéralogie*, p. 21. (Masson. Paris, 1899).

parallel to themselves by sliding the length of one of their ranges, *t*, and this can take place without causing rupture. The reticulated plane, *P*, conjugate of *T* and passing by the range, *t*, is called the sliding-plane (*plan de poussée;* in German, *Schiebungsebene*).

Let us suppose that the plane of translation and the sliding-plane are rectangular, and that the sliding-plane is a plane of symmetry of the system. Let *A B C D* (Fig. 2) be a mesh of this sliding-plane, *A B* and *C D* being parts of two immediately adjoining planes of translation. If a movement of translation takes place, while *C D* remains fixed, *A B* could slide on itself.

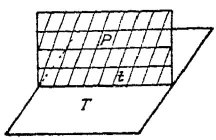

FIG. 1.—PLANE OF TRANSLATION AND SLIDING-PLANE.

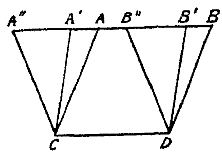

FIG. 2.—MOVEMENTS OF TRANSLATION.

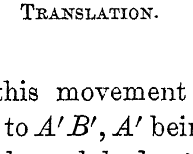

FIG. 3.—SLIDING-PLANE BEFORE TRANSLATION.

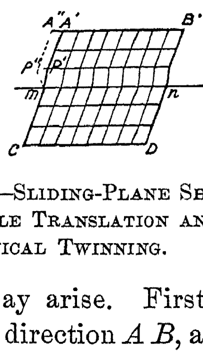

FIG. 4.—SLIDING-PLANE SHOWING SIMPLE TRANSLATION AND MECHANICAL TWINNING.

In this movement two conditions may arise. First, *A B* moves to *A′B′*, *A′* being any point in the direction *A B*, and the molecular polyhedra, of which the centers of gravity coincide with the intersections *A* and *B*, retain their initial orientation after displacement. There is only a change in the form of the mesh *A B C D*, which is now in a condition of unstable equilibrium. This would be called a simple translation. Second, *A B* moves to *A″ B″*, in a position symmetrical with its original position, and, at the same time, the displaced molecular polyhedra also assume the orientation symmetrical with their first orientation, with reference to a plane perpendicular to the direction of translation. In this case mechanical twinning ensues.

The translation, with or without the formation of twins, gives place to the appearance of little straight steps on the exterior faces of the crystal, and these steps are parallel to the lines of the planes of translation. If $A\ B\ C\ D$ (Fig. 3) be a portion of the sliding-plane, let us suppose that, the part $C\ D\ m\ n$ remaining stationary, the plane of translation immediately adjoining $m\ n$ is displaced for a short distance, involving the upper part of the system. The terminal lines $A\ C$, $B\ D$ will take the profile $C\ m\ p'A'$ (Fig. 4) if the translation is simple, or $C m p''A''$ if there is mechanical twinning.

To distinguish a line of translation from a mechanical twin crystal, the deformed face must be re-polished and etched by a suitable reagent. Lines of translation are not in evidence, because the molecular polyhedra have retained their orientation; the macle, on the other hand, shows up—it has a definite thickness, its planes of junction are depresssed, and its corrosion figures are not the same as those of the initial crystal, since the orientation of the molecular polyhedra has been modified.

The notion of translation can easily be generalized. It is, in fact, probable that the so-called planes of translation are only the planes of easiest translation, and that when the faculty of deformation in the direction of these planes is exhausted, others enter into action in their turn. Imagine, then, that the mesh, $A\ B\ C\ D$ (Fig. 2), after having been transformed by simple translation from one parallelogram into another, susceptible of conserving an unstable equilibrium, can, by other movements of translation, change into a quadrilateral—the only condition being that the area remains constant, inasmuch as the density does not change. Ultimately, when all the meshes of a crystalline body are dislocated in this manner, it becomes in a way decrystallized, and this explains the curved deformations (folds, or *plissements*).

These views on deformation of crystals, still comparatively little known, appear to have originated from the celebrated experiments of Reusch and Baumhauer, on the mechanical twinning of calcite. Simple translation was introduced into science by a long series of notes due to Prof. Mügge, and published for the most part in the *Neues Jahrbuch für Mineralogie*. Prof. Mügge's researches date back to 1884 at least, but his first work was on minerals, and it was only in 1899 that he took up

native metals.[5] The same year Prof. Ewing and Mr. Rosenhain undertook, independently, the study of the industrial metals, and in a remarkable memoir,[6] which has been followed by many more, were developing from their side the idea of translation. In the preceding considerations we have only advanced a tentative personal interpretation, ascribing to a single cause lines of translation, twinning, and folding. If we do not retain, in this instance, the term "slip-bands" proposed by Prof. Ewing and Mr. Rosenhain, it is because this term includes both the lines of translation and the folds, between which we consider it useful to make a distinction.

In the case of bodies belonging to the same crystallographical system, the position of the planes of translation and of mechanical twinning, and also the presence or the absence of folds, furnish many very valuable differentiation characters.

2. *Congenital Twinning.*

We give this name to the twinning which takes place spontaneously during the progress of crystallization, just when the solid molecules are becoming isolated from the liquid medium which holds them in a state of fusion or in solution.

3. *Twinning Resulting from Annealing after Deformation.*

This twinning results, as the name indicates, from the annealing, for a sufficiently long time and at a sufficiently elevated temperature, of a crystalline solid previously deformed.

This twinning might be susceptible of relation with mechanical twinning and the phenomena of translation. If, for example, translation has moved the point A (Fig. 2) to the vicinity of the symmetrical point A'', but without the molecular polyhedron, which has its center of gravity at A'', having assumed the orientation corresponding to the new position of the system, annealing, when widening the molecular intervals, could render the molecular polyhedron A free to assume the position of twinning equilibrium, which, as a result of the deformation, the system alone had taken, or nearly so. The twinning will only then be consummated. Provided matters actually pro-

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[5] *Neues Jahrbuch für Mineralogie*, vol. ii., p. 55 (1899).

[6] *Transactions of the Royal Society*, vol. cxcv., pp. 279 to 301 (1900).

ceed in this fashion, the annealing simply completes the work of deformation.

4. *Mechanical Properties Functional of the Crystalline Orientation.*

The possible variations in these properties result from the very definition of the crystalline structure. It has been a known fact for a long time that the cleavage-faces are faces of minimum hardness, and that on the same face the hardness varies with the direction and along the same direction with the sense of the striation.

5. *Synchronous Crystalline Figures.*

When a body is caused to crystallize on a crystalline face of another body, it occasionally happens that the structure of the latter orients the molecules of the crystallizing body to the extent that, as they separate from the bath, it imposes upon them a pseudosymmetry which does not naturally belong to them. These anomalies[7] can eventually give certain indications concerning the crystallography of the dominant body.

6. *Segregation Figures.*

When a body, liquid or solid, deposits several successive solid phases, the deposits of the second or third consolidation frequently settle by preference between certain definite crystallographic planes of the deposit from the first consolidation, and in this way show up its structure.

These planes between which the segregation takes place are probably the planes of maximum separation, or, which comes to the same thing, planes of the greatest reticular density.

II. Experimental Section.

Such are the methods we have used or attempted to use.

All of them, of course, have not furnished results which could be used for the end we had in view—that is to say, means of diagnosis applicable to the different varieties of iron. Whether by reason of their inappropriateness, or because we did not know how to make use of them, certain of them have

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[7] See an article by M. Wallerant : *Bulletin de la Société française de Minéralogie,* Aimée, p. 180 (1902).

given negative results. This will not deter us from describing any facts observed, whether positive or negative, which could suggest ideas for fresh experiments to other workers.

We must evidently work for each of the allotropic varieties of iron within those limits of temperature where the particular variety is stable.

For α-iron there is no difficulty, since ordinary temperature is within the range of stability.

For β-iron, which cannot be wholly kept in unstable equilibrium, temperatures between 750° and 855° C. (we retain the temperatures indicated in our preceding publications. To take into account the new pyrometric standards, these figures must be raised to about 780° and 890° respectively—that is to say, to the figures given by Roberts-Austen or by Carpenter) are required, as far as possible about the middle of the interval, in the neighborhood of 800°. Besides, a crystal of α-iron heated in the range of β-iron, and cooled to the ordinary temperature, becomes again the same crystal of α-iron—the system persists beyond transformation $A2$.

This is not the case with the passage of the point $A3$, provided it has been sufficiently exceeded, and for a sufficiently long time. It is quite possible, if the heating and the cooling down are both done rapidly, to heat a crystal of α-iron up to 900° without destroying it; but if it is maintained at this temperature, the crystal resolves itself into little grains, with the formation of elongated lamellæ, which appear to be twin crystals. Fig. 5 is a photomicrograph (100 diameters) of face $p(001)$ of a crystal of iron, after two hours' heating at about 1,000°; etched, after re-polishing, by alcoholic picric acid; the sides of the photomicrograph are parallel to the sides of the square of the original crystal.

It is therefore preferable for the study of the crystallography of γ-iron to resort to the alloys of iron with carbon and manganese, nickel or chromium; naturally selecting from these alloys those that are not magnetic at the ordinary temperature.

In all cases it is expedient to have the crystals as large as possible.

As regards α- and β-irons, thanks to the generosity of Professor Tschernoff, we had at our disposal a beautiful specimen

from an open-hearth furnace, with large cubic cleavages, containing C, 0.05; P, 0.30; Mn, 0.00 per cent.

Another sample, still more remarkable, was procured for us by Mr. Werth, director of the metallurgical works of Denain and Anzin. It consists of the fragments of an old steel rail, which during 15 years had served as a guide to a damper in a furnace-chimney, and of which certain parts had during this long period of time been submitted to conditions favorable to the development of crystals. That is to say, in accordance with Mr. Stead's excellent work,[8] to a temperature slightly inferior

FIG. 5.—FACE $p(001)$ OF A CRYSTAL OF IRON HEATED FOR TWO HOURS TO 1,000° C., POLISHED AND ETCHED.

to *A*2. Chemical analysis of the sample made in the laboratory at Denain yielded C, 0.06; Si, 0.05; S, 0.02; P, 0.116; Mn, 0.30 per cent. Yet the amount of carbon found must be above the mean, inasmuch as we have encountered no traces of cementite in the numerous sections we have made, and the other oxidizable elements are to a great extent scorified away, so that the metal is almost pure iron, in crystals, of which some attain a magnitude of several cubic centimeters.

For γ-iron we have used a fragment of ordinary cast man-

[8] *Journal of the Iron and Steel Institute*, vol. liii., No. 1, pp. 145 to 189 (1898).

ganese steel, which Mr. Hadfield kindly had taken from the interior of an ingot, in the region of final consolidation. Another sample came to us from the Imphy Steel Works, and contained C, 0.15; Si, 0.30; P, 0.023; Mn, 0.23; Ni, 24.80; Cr, 2.21 per cent. Although special precautions had been taken to delay the cooling, this sample, which was in the form of a round bar, presented a comparatively fine grain inappropriate

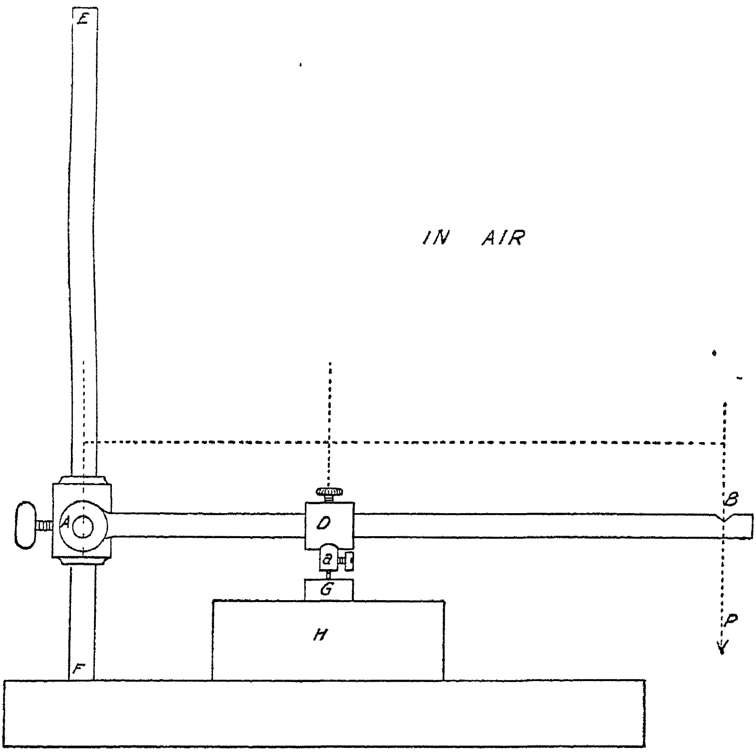

FIG. 6.—DEFORMATION-APPARATUS FOR USE IN AIR.

for crystallographic researches. We deformed a piece of it which was afterwards annealed at about 1,300°, and obtained equi-axial grains of a mean diameter of 1 mm. Mr. Hadfield's manganese steel, on the contrary, showed on fracture distinct crystallites with rectangular branches. It is known that the axes of a crystallite in the cubic system are the quaternary axes of the cube; consequently it was possible to cut sections with a known crystallographical orientation.

1 (*a*). *Continuous Crystalline Deformation; Silhouettes.*

The principle of the method has been described above. A sewing-needle served for the tests at the ordinary and at lower temperatures; it was broken, and a point perfectly acute, with an angle of about 60°, was re-made at the hardest part by rubbing on emery-papers of increasing degrees of fineness.

This needle, *a* (Fig. 6), is fixed by a vise in a socket, *D*, which can slide along the lever, *A B*, and be fixed at any point in its length. The lever, *A B*, is articulated at *A* on a movable guide along the vertical arm, *E F*. To obtain a silhouette by the pressure of the point on a polished crystallographical face, the prepared specimen, *G*, is placed on the support, *H*, in such a way that the polished face is horizontal. *A* is moved the length of *E F* until the lever, *A B*, is also horizontal and the point just meets the piece *G*. Then suitable weights are sus-

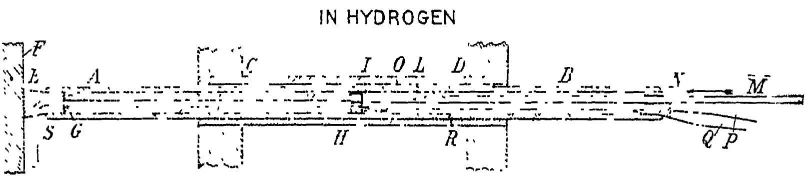

FIG. 7.—DEFORMATION-APPARATUS FOR USE IN HYDROGEN.

pended from *B* so that, taking into consideration the length of the arms of the lever, a desired pressure is brought on *a*.

Between 200° and 400° C. the ordinary needle would lose its hardness; so it is replaced by a needle made of high-speed steel.

Above 400° C. it is no longer possible to work in the open air; the oxidized film obliterates the silhouettes and the lines. The experiment must be made in hydrogen or nitrogen, and the needle made from cast quartz.

A porcelain tube, *A B* (Fig. 7), is heated along *C D* in a Mermet furnace. The extremity, *A*, is closed by a cork, *E*, which abuts against a wall, *F*. Another porcelain tube, *G H*, with an external diameter rather smaller than the internal diameter of *A B*, rests against the cork, *E*, at one end, while at the other end the specimen to be tested, *I*, is placed, along with two pieces, *K*, of the same metal, which press between them the end of a Le Chatelier couple.

The pieces, *I* and *K*, form together a cylinder which passes

into the tube without friction. The quartz needle, *L*, is set in an iron tube, *M*, passing with slight friction the cork on the right, *N*, and resting on a partition, *O*, of thin sheet-iron diametrically placed. The wires of the couple, *P*, *Q*, pass between the tube and the cork, *N*, and are separated from one another within the tube by a pipe-clay tube.

The holder of the needle serves also as the gas-inlet, and is perforated with a hole, *R*, for the purpose. To make a test, the apparatus is cleared out by means of pure dry hydrogen, entering at *R*, and passing out at the tube, *S*, which penetrates through the cork, *E*, and is bent at a right angle. When this is done, *S* is closed, because the play between the handle of the needle and the cork, *N*, is too great to insure a fast joint, and it is by this passage that the gas now escapes. The furnace is lighted, and the temperature raised very slowly to the desired degree. When this is attained the needle is pushed against the polished surface, *I*, and pressed against it. With a little practice the pressure is easily regulated, so as to obtain impressions that are neither too large nor too small.

The same set of apparatus could serve for nitrogen were it not that the difficulty of drying nitrogen is greater than in the case of hydrogen; because tubes of sodium, which give off hydrogen as a result of the decomposition of the water-vapor, are not available, and to keep a polished surface of iron intact the desiccation must be perfect. We have therefore modified the apparatus in the following manner:

Two similar iron tubes, *A* and *B* (Fig. 8), closed at the lower end, are placed upright side by side in a nickel crucible, *C*. The test-piece, *D*, is placed at the bottom of tube, *A*, and covered with a convex disk, *E*, of thin sheet-iron, with a small hole in the center, which supports the point of the needle, *F*. At the bottom of tube, *B*, there is a piece of iron, *G*, in which is inserted the end of a Le Chatelier couple, the wires from which, *H*, *I*, are separated from one another by a pipe-clay tube, and from the iron tube by a cylinder of mica. In making an experiment the crucible is filled with iron-turnings, which regulate the temperature, while *A* is filled up between the tube and the holder of the needle with iron which has been reduced by hydrogen at an incipient red heat; the whole is heated over a Méker burner until the desired temperature is attained. This

temperature is maintained, and the holder, K, of the needle is pressed. The pressure-figure is made, and there is nothing more to do but to let the test-piece, D, cool before removing it.

The iron should be reduced by hydrogen at a temperature sufficiently high for it not to be pyrophoric, and yet just as low as possible. It then serves to absorb the small quantity of oxygen retained at the bottom of the tube, and also any that may leak in during the progress of the operation. Its use was suggested to us by Mr. Lebeau, and it has succeeded perfectly. The compact iron is less prone to oxidation, and is protected by it. At first we tried copper, which is, in fact, protected by the iron; then we tried Goldschmidt manganese, which arrests the oxidation of iron quite well, but affects its surface, probably on account of the presence of impurities.

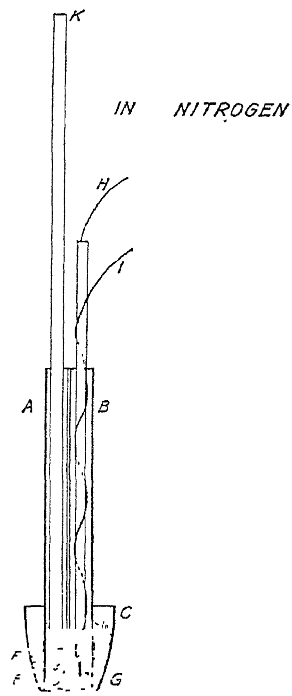

Fig. 8.—Deformation-Apparatus for Use in Nitrogen.

According to circumstances, one or other of the three apparatus mentioned has been used.

On α-iron the indenting has been done at the temperature of liquid air, at the ordinary temperature, at blue temper, and at 600° C.

On β-iron at about 800° C.

On γ-iron at 900°, and when employing manganese or nickel steels, at the ordinary temperature.

In every case the silhouettes have the same form on the same crystallographic face, whether the iron was in the state α, the state β, or the γ-state.

On the cube face $p(001)$ it is a cross, of which the arms are parallel to the diagonals of the square, and which has four axes of symmetry parallel respectively to these diagonals and to the sides of the square.

On a rhombododecahedral face $b^1(011)$ the figure may still have four arms, but not rectangular. Most frequently these branches coalesce in pairs, and give a silhouette of two sheets

turned towards the faces of the cube which are perpendicular to the section under consideration. The figure has only two axes of symmetry, parallel respectively to the sides of the rectangle.

On an octahedral face a^1(111) there is a figure with three lobes, of which the axes of symmetry are the heights (or bisections) of the equilateral triangle.

To represent these silhouettes, let us imagine a cleavage-cube which has been subject to the truncations b^1 and a^1, apply these truncations to the face of the cube, which they cut at

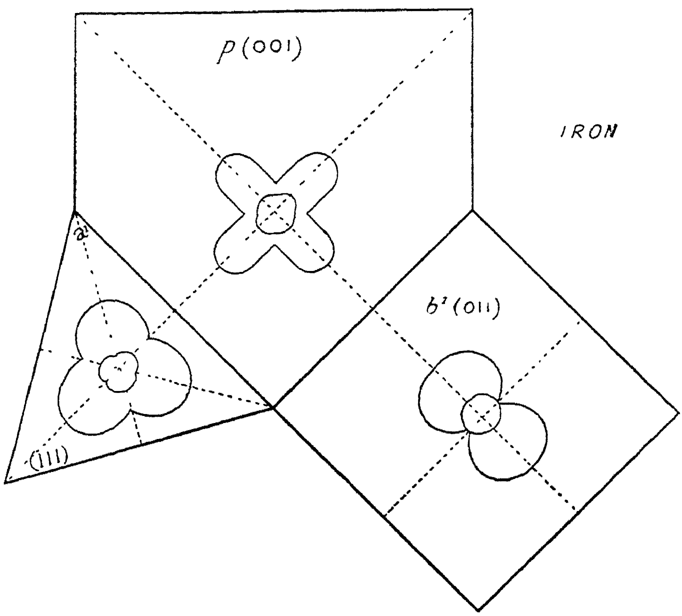

FIG. 9.—PRESSURE-FIGURES.

45°, and project the whole on the plan of the illustration. We shall thus have Fig. 9, upon which the silhouettes that we have just described are drawn.

These results confirm the conclusion that we have drawn from our previous trials—that is to say, that the three allotropic varieties of iron belong to the cubic system. In fact, this is the only system that can give figures with four axes of symmetry on three rectangular faces. The quadratic system could only have these on the bases p(001) parallel with one another, and the cross, with four axes of symmetry, obtained fortuitously on one face, would not be repeated on an adjacent

rectangular face, as we have ascertained does happen. In the rhombohedric system the symmetry on one face $p(100)$ is, of course, very poor. Hence, with bismuth, which crystallizes in rhombohedra, simulating cubes, the pressure-figure on a polished natural face is that shown in Fig. 13 (75 diameters). There is only one axis of symmetry, parallel to one of the diagonals of the rhomb. Perpendicular to this axis there are the twin crystals or the lines of translation which have been described by Prof. Mügge.[9]

Of course our conclusions assume that iron remains in the γ-state from the point $A3$ up to fusion. Should it be demonstrated that the point of Ball and Curie, about 1,300°, corresponds to an allotropic transformation, it would be necessary to split γ-iron into two and to revise matters.

1 (*b*). *Discontinuous and Effaceable Deformation Figures.*

Every mode of deformation can be employed to produce these lines. But we have principally resorted to that which has already served us in obtaining the silhouettes described in the preceding section—that is to say, the normal pressure of a needle.

α-Iron.—The crystals are sliced from the cleavage-faces.

We shall first describe as typical the tests made at the ordinary temperature. The results are assembled in Fig. 10. The weight on the needle was 1.6 kilograms.

On face $p(001)$ the branches of the cross are formed of the folds $c\ d\ e\ f$, which envelop one another. The portions $c\ d$, $e\ f$ are nearly parallel to the diagonals of the square, and might be lines of translation; they are connected together by an approximately semicircular arc. Sometimes, from some unknown cause, this arc is replaced by a line, $g\ h$, which is comparatively straight and parallel to one diagonal of the square, and is connected by small arcs to the lines $h\ i$, $g\ f$.

On the truncation $b^1(011)$ the figure as a whole is still a cross, but its arms, $X\ X$, $Y\ Y$, are no longer rectangular. The acute angles, $X\ O\ Y$, are turned towards the intersection of the truncation with the face of the cube which is perpendicular to it. The arms are formed of the folds, $c\ d\ e$, roughly elliptical, and

[9] *Neues Jahrbuch für Mineralogie*, vol. i., pp. 183 to 191 (1886).

enveloping one another. Almost always these folds, instead of closing up, coalesce on the bisection of the acute angles, either after inflection, as represented at $f g$, or without inflection, as at $k i h$. No straight line is visible.

On truncation $a^1(111)$, one can observe on each of the three axes of symmetry: firstly, nearly straight lines, $c d$, which are situated between the point of impact and a summit of the triangle, and parallel to the side opposed to this summit; secondly, curved lines, $e f$, which part at e from one of the axes of symmetry between the point of impact and the side of the

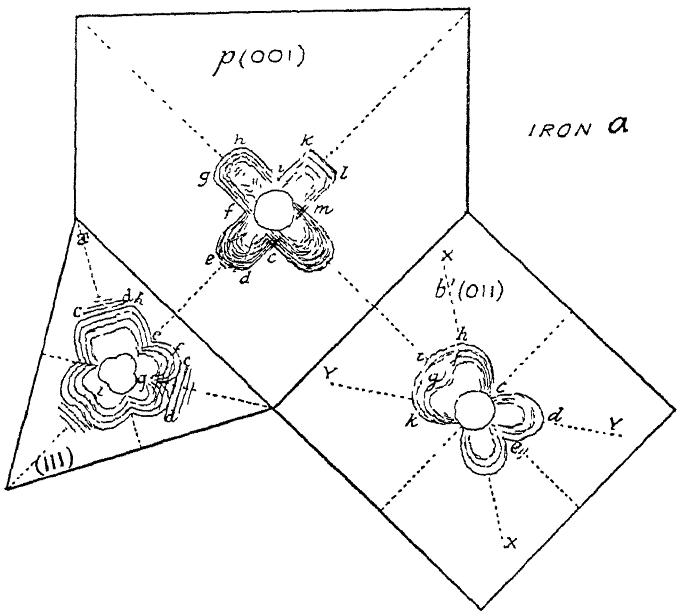

FIG. 10.—PRESSURE-FIGURES ON α-IRON.

triangle normal to the axis concerned. These lines, $e f$, have at their origin the aspect of spirals; they may bend, following $g f$ towards the adjacent axis of symmetry, but more frequently they coalesce, as at h, with the straight lines $c d$, or again with their neighbors after inflection, as at i, or without inflection.

Other tests have been made on face $p(001)$ at various temperatures. The silhouettes are the same in all cases, but the details may show some variation.

At the temperature of liquid air, under charges of 1.5 to 3 kg., the silhouettes are nearly without details.

At a blue temper heat there were produced many times one or more branches, such as *i k l m* (Fig. 10; face *p*)—that is to say, formed exclusively of straight lines; but these could not be reproduced at will.

At about 600° C. in hydrogen, the figure is the same as in the cold, but more subdued, and with little detail.

β-Iron.—It seems that at a red heat in hydrogen the polished iron surface suffers some modification, and that a superficial skin forms which masks the details of the deformation; even

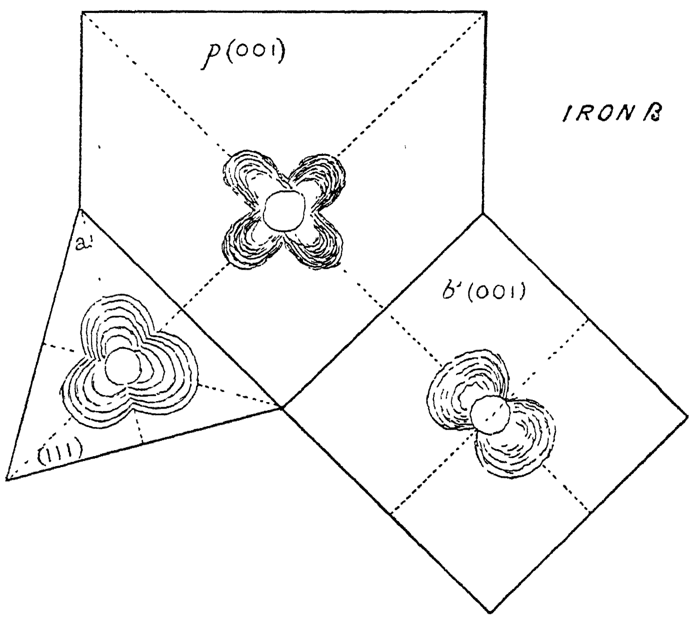

FIG. 11.—PRESSURE-FIGURES ON β-IRON.

after cooling, fresh indentings made at the ordinary temperature show nothing more than the silhouette. It is true that the details may be made to appear later on, either by spontaneous oxidation in the air (Fig. 14; 150 diameters; face *p*), or by etching with picric acid; but much sharper results are obtained by working in nitrogen with the apparatus (Fig. 8).

On face *p*(001) the arms of the cross are exclusively formed of folds which cover or envelop one another; these folds appear to be generally more rounded than those of *α*-iron.

On face b^1(011) the figure is the same as that for *α*-iron.

On face a^1(111) the straight lines seen with *α*-iron are no

longer observed. The curved lines similar to the contours of the silhouette alone remain. See Fig. 11.

γ-Iron.—On face $p(001)$ the lines which cover the silhouette of the cross are straight, and parallel to the diagonals of the square.

On face $b^1(011)$ the lines are again straight, and belong to three systems—one is parallel to the intersection of the truncation with the face of the cube which is normal; the other two are symmetrical in relation to the sides of the rectangle, and make with one another, with a very gratifying approximation,

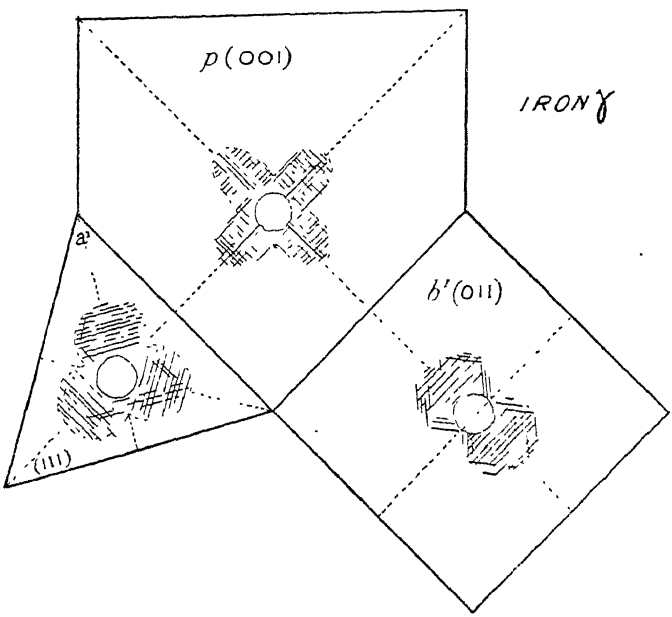

FIG. 12.—PRESSURE-FIGURES ON γ-IRON.

the theoretical angle of the faces of the octahedron—that is, 109° 28′.

In face $a^1(111)$ the lines are always straight, and parallel respectively to the three sides of the equilateral triangle.

The results are arranged on Fig. 12.

We have also made deformation tests on practically pure iron above 900°. Under these conditions, we have seen a single crystal break up into grains; but under the condition of working in hydrogen, there are on these little grains lines of deformation; they are still straight lines—lines of translation.

The curved lines of β-iron and α-iron are therefore not attributable to a question of temperature. (We learn, subsequent to writing this note, that Mr. Rosenhain has made observations on γ-iron which agree with ours, and which were presented at the meeting of the Iron and Steel Institute in May.)

If we compare the three varieties of iron, we see at the start that the lines of deformation are exclusively straight on γ-iron. γ-iron, hence, has planes of easy translation, and according to directions noted, these planes are the planes

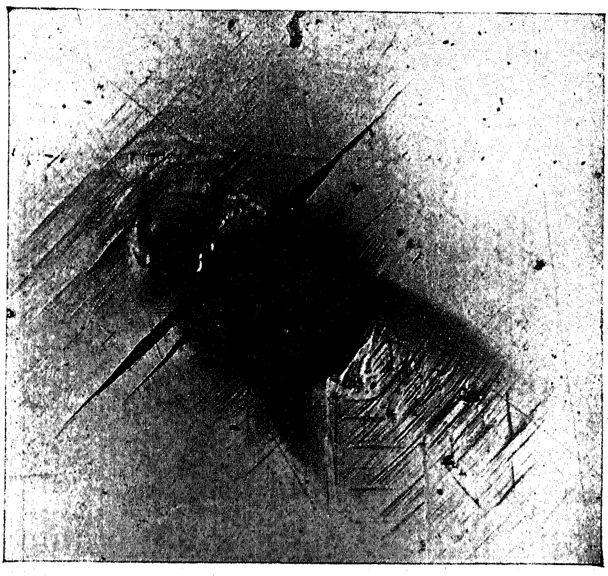

FIG. 13.—PRESSURE-FIGURE ON BISMUTH.

$a^1(111)$—that is to say, parallel to the faces of the octahedron, exactly as in the case of copper, gold, silver (Mügge), and lead (Humphrey).[10]

On β-iron the lines are exclusively curved; there are no planes of translation.

On α-iron there is, at least on faces p, and especially on faces a^1, a mixture of straight and curved lines. The orientation of the straight lines apparently proves the existence of planes of translation parallel with the faces of the octahedron—planes also referred to by Messrs. Ewing and Rosenhain,[11] and of

[10] *The Metallographist*, vol. vi., pp. 250 to 258 (1903). [11] *Loc. cit.*

which we had previously disputed the presence in iron. But the curved lines dominate considerably, therefore the translation is difficult, and the greater part of the deformation seems due to another mechanism.

We have employed on α-iron other modes of deformation—tensile tests, compression, bending—and in every case we have found scarcely anything but curved foldings, without any relation to the crystallographic directions either as a whole or in their details. We shall cite only one experiment, which appears to us convincing. We impressed the pressure-figures—that

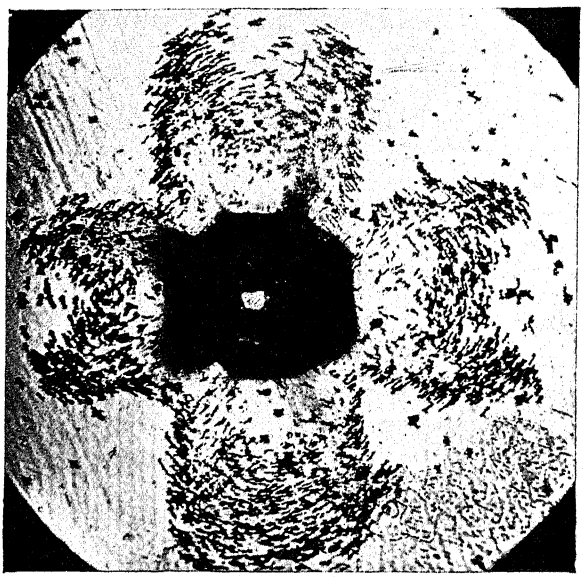

FIG. 14.—PRESSURE-FIGURE ON β-IRON AFTER OXIDATION.

is to say, the cross of Fig. 10—upon a crystal cut into an elongated rectangular sample, and presenting on two lateral faces the faces of the cube; the figures indicated immediately the crystalline orientation. Then we submitted the bar to gentle bending a little beyond the limit of elasticity. This new deformation produced new lines in the vicinity of the crosses, and on the crosses themselves; Fig. 15 (200 diameters) shows one of the crosses and its surroundings. It will be noticed that the lines produced by the bending are neither parallel to the diagonals of the square (axes of the arms of the cross) nor

to the sides of the square, as they ought to be, if they were the lines of translation following the faces of the octahedron or of the cube.

1 (*c*). *Non-Effaceable Deformation Figures: Mechanical Twinning.*

The lines that have just engaged our attention, lines of translation or foldings, if obliterated by re-polishing, do not reappear under the influence of etching. And, to our knowledge, slight static deformations produce no others.

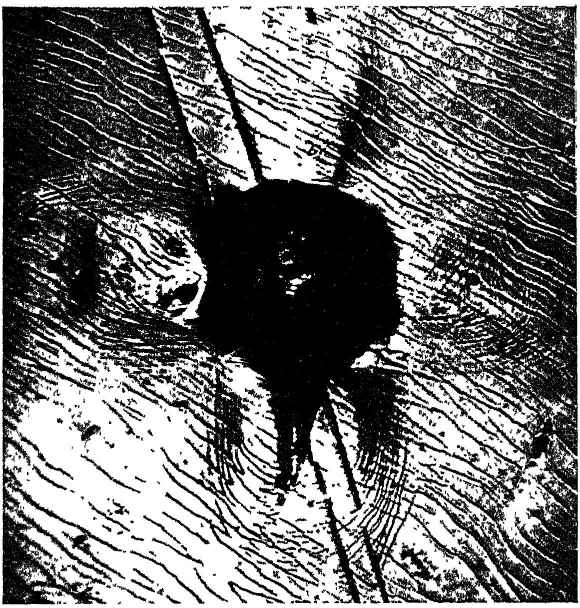

FIG. 15.—LINES PRODUCED BY BENDING.

On the other hand, under certain circumstances that we are going to examine, lines are obtained that etching reveals after re-polishing. These are actual lamellæ, with very slight but decided thickness; in other words, mechanical twin crystals or macles.

a-Iron.—The lamellæ known as Neumann's lamellæ or lines, in honor of the savant who discovered them in 1850, have been recognized for a long time in cubical meteoric iron and in kamacite (nickel ferrite). Their presence in terrestrial iron was noted by Prestel. (For the history of Neumann's

lines we have drawn, to a large extent, on the excellent book of Professor Cohen, *Meteoritenkunde*, Stuttgart, 1894.)

Neumann's lamellæ are visible frequently to the naked eye, on a cleavage-face. On these faces they are parallel either to the diagonals of the square or to the lines which join the angles at the center of the opposed edges (Fig. 16).

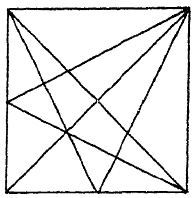

FIG. 16.—NEUMANN'S LAMELLÆ.

After polishing to low relief, they appear as slight depressions on the faces of the cube, sometimes as depressions, sometimes in relief upon any other crystallographic section. The best reagents for showing that they have a decided thickness are alcoholic picric acid and nitric acid, the latter very dilute (1 in 500), which gently eats away the planes of junction (Figs. 17, 250 diameters, and 18, 600 diameters). With reagents susceptible of giving corrosion figures, the interior of the lamellæ can assume a different color from that of the principal crystal which contains them.

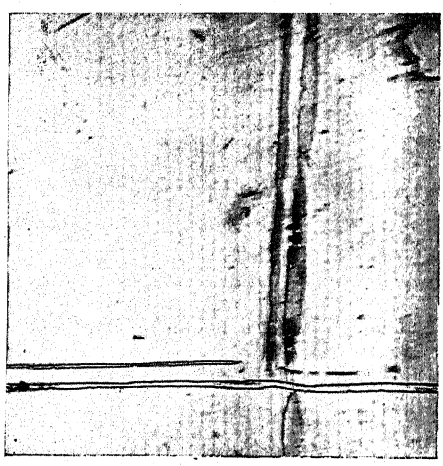

FIG. 17.—NEUMANN'S LAMELLÆ AFTER ETCHING.

The lamellæ may be of uniform thickness, like those to be seen parallel to the sides of Fig. 17, or present a more or less regular indentation, probably connected with the formation of another system of lamellæ (Fig. 18). Fig. 19 (250 diameters; etched with picric acid) is a good example of indented or jagged lines on a non-oriented section.

If the etching is vigorous—for example, with 10 per cent. nitric acid for half a minute, after a previous etching for four minutes with double chloride of copper and ammonium to show the corrosion figures—Neumann's lines are then very much en-

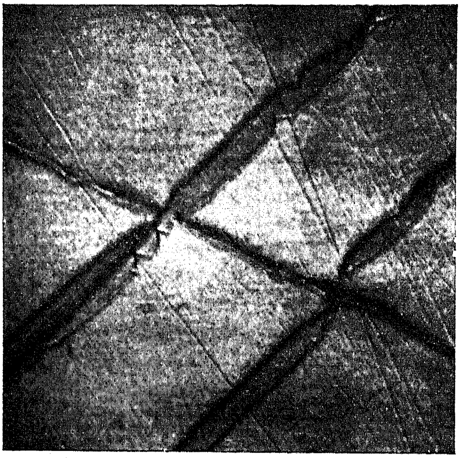

FIG. 18.—INDENTED LAMELLÆ.

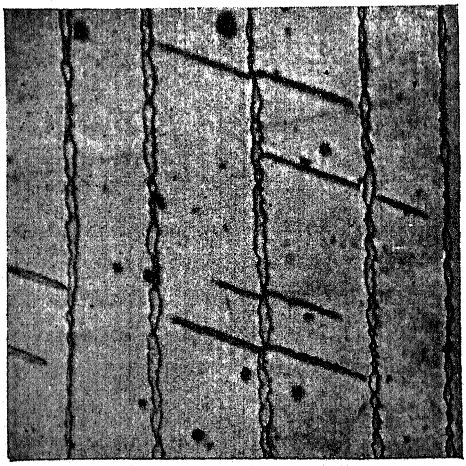

FIG. 19.—INDENTED LAMELLÆ.

larged; they become visible like so many fine file teeth, and if they are numerous enough, give the piece a watered aspect. Seen through the microscope, the line is channeled (Fig. 20; 250 diameters), but these channels, corrosion figures, markings

of strong etching, extend really to the edges of the lamellæ, and it becomes almost impossible to distinguish the lamella itself from the corrosion figures belonging to it.

Neumann's lamellæ have given rise to numerous works. It is agreed to regard them as twin crystals; but many divergent opinions have been expressed on the nature of these twins, on their position in the principal mass, and on the law of twinning.

Neumann[12] and, later, Tschermak[13] admit that it is a question of fluorspar twinning, which is represented (Fig. 21, 1*a* and 1*b*) in elevation and plan; the octahedral face is the plane of

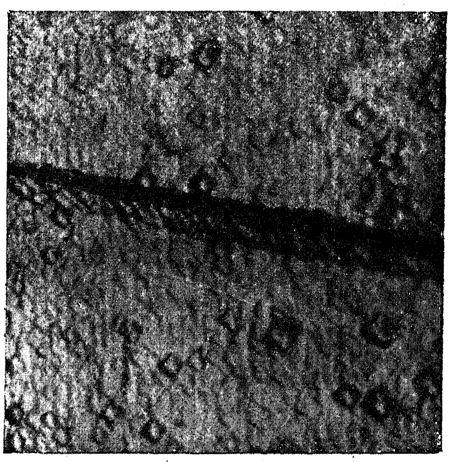

FIG. 20.—PHOTOMICROGRAPH SHOWING CHANNELING.

twinning, and the ternary axis the axis of twinning. As there are four ternary axes, the lamellæ could belong to four cubic sub-elements imbricated in the dominant cube; and the twenty-four faces of these four sub-elements, identified with the planes of Neumann, would consequently be parallel respectively to the faces of the trioctahedron $a\frac{1}{2}(122)$. Fig. 21, 1*c*, indicates the position corresponding to a lamella on a diametrical plane passing through two edges of the dominant cube; the face of junction is the face of the cube of one of the sub-elements.

Rose,[14] who only studied the direction of etched lines,

[12] From Cohen.

[13] *Sitzungsberichte der Akad. d. Wiss. zu Wien, Math. Nat. Classe*, vol. lxx., p. 443 (1874).

[14] From Cohen.

could not decide whether the lamellæ are parallel to $a\frac{1}{2}(122)$ or to $a^2(112)$.

Sadebeck[15] has observed, on a cleavage-face of the dominant cube, that the faces terminal to those of the lamellæ, which make an angle of 45° with the sides of the square, make an angle of $144\frac{3}{4}°$ with the plane of the face of the cube. Regarding these terminal faces as the cleavages of the sub-ele-

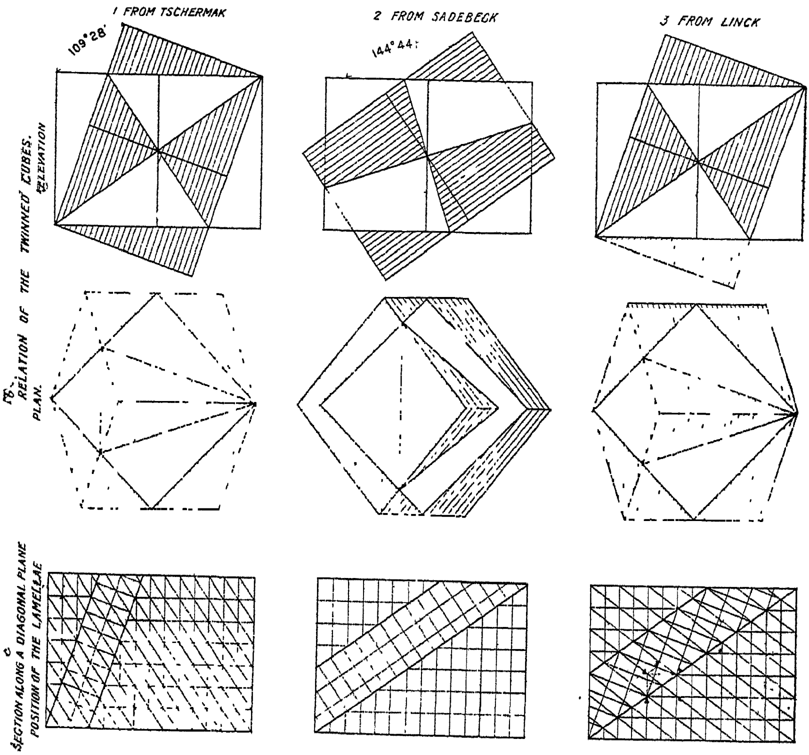

FIG. 21.—NEUMANN'S LAMELLÆ.

ments, he concluded that the plan of twinning belonged to a trioctahedron (20.20.9)—the axis of twinning being perpendicular to a face of this trioctahedron. The law of association, represented by Fig. 21, 2, thus would become very simple; for in accordance with this law, and in the case of the two elements twinned, if one considers two faces of the twinned cubes cutting one another in the direction of a common diagonal, the face of one of these cubes is a face of the trapezohedron (112)

15 *Poggendorff's Annalen*, vol. clvi., pp. 554 to 563 (1875).

of the other; moreover, in the same zone, an octahedral face of the dominant cube is the rhombododecahedral face of the secondary cube.

Finally, Linck[16] admits, with Neumann and Tschermak, the law of fluorspar twinning, but the junction faces should be $a^2(112)$ and not $a^{\frac{1}{2}}(122)$. See Fig. 21, 3*c*. Under these conditions the planes of junction have the same notation for the two unit elements twinned. The planes $a^2(112)$ are at the same time planes of translation: the summit *d* of the mesh *a d b c* (Fig. 21, 3*c*) of the dominant cube has only to be transported at *d′*, parallel to *a b*, in order to form the twin.

In face of these different opinions, fresh researches would apparently not be useless.

Taking a trihedral cleavage-angle on Professor Tschernoff's iron, we cut a rectangular parallelopiped measuring about 15 by 15 by 7 mm., consisting mainly of a single crystal. We polished, etched, and photographed the six faces successively, and stuck the photographs on a wooden model of the size desired. The lamellæ of Neumann could in this way be followed on two or more faces; they were very numerous in the specimen, and it was certain that they were always parallel to the planes $a^2(112)$, which contradicts Tschermak's law of junction, and confirms that of Sadebeck and of Linck. The law of twinning still remained to be decided. It appeared difficult to decide experimentally between the two which have been proposed: Neumann's lamellæ are far too thin to permit of them taking pressure-figures; and for the same reason one would not fare better in attempting interpretable corrosion figures. But it may be remarked that Sadebeck's twin would be unique of its kind. The observation which suggests this—that is to say, the angle of $144\frac{3}{4}°$ made by the fracture-facets of the lamellæ with the plane of cleavage of the dominant cube—is easily explained by Linck's conception: these fracture-facets are just simply the planes (112) of the lamellæ; and an experiment made on the

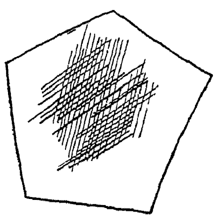

FIG. 22.—MECHANICAL TWINNING AND LINES OF TRANSLATION.

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16 *Zeitschrift für Krystallographie*, vol. xx., p. 209 (1892).

principal dominant cube shows actually that the planes of junction (112) of Neumann's lamellæ are also the possible planes of fracture. Therefore Linck's theory seems to us sufficiently demonstrated.

The question whether Neumann's lamellæ are congenital twin crystals, as Tschermak thought, or the result of mechanical twinning, in accordance with the common view of Sadebeck and Linck, remains to be decided. If we have to deal with congenital twinning, or a product formed during solidification, the twin crystals would really belong to γ-iron; but as we do not depart from the cubic system, the system would be maintained throughout the transformations, and a twin of γ-iron remain a twin of α-iron. There is, therefore, no objection on this head. But congenital twins are of the same magnitude as the crystals from which they are derived, since the development of the two elements twinned has been simultaneous. Neumann's lamellæ, on the contrary, are extremely small; moreover, as we have seen (Figs. 18 and 19), they are frequently inflected and thrust aside by meeting lamellæ of another system. Linck has also observed in the meteoric iron of Braunau that the delicate lamellæ of rhabdite, $(Fe, Ni)_3P$, are broken and thrust aside by the lamellæ of Neumann; at least, it is easy to produce these lamellæ artificially by deforming, by shock, a fragment of crystal, of which the pre-existing lamellæ have been recorded by a photograph. Notably when a crystal of iron is cleaved, Neumann's lamellæ appear in abundance on both sides of the cleavage-face.

Therefore there is no doubt that Neumann's lamellæ are mechanical twins. Up to the present we have only been able to obtain them by shock, and more readily the lower the temperature. The Swedish iron that Mr. Hadfield ruptured by traction at the temperature of liquid air, exhibits many of them in the vicinity of the fracture.[17] At the ordinary temperature, the production of lamellæ is still easy, at least by shock, as we have already said. But it does not take place at the temperature of blue temper, nor at higher temperatures.

β-Iron.—In this case there are no known mechanical twins.

γ-Iron.—Slight deformation, without shock, only gives effaceable lines of translation. But more severe deformation fur-

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Journal of the Iron and Steel Institute, vol. lxvii., No. 1, p. 248 (1905).

nishes twins as well. On our nickel-chromium steel from Imphy we have impressed marks deeply in the cold; the surface marked in this manner was partly filed, re-polished, and etched with a hydrochloric solution of iron perchloride. The etching shows, in the region of most deformation, parallel double lines, with the space between of different coloration from that of the grains in which they occur—this is therefore a question of twins. To determine the orientation, the etched piece was subjected to slight general deformation; lines of translation appeared, and it was always observed that one of the systems of lines was alongside or parallel with the twins

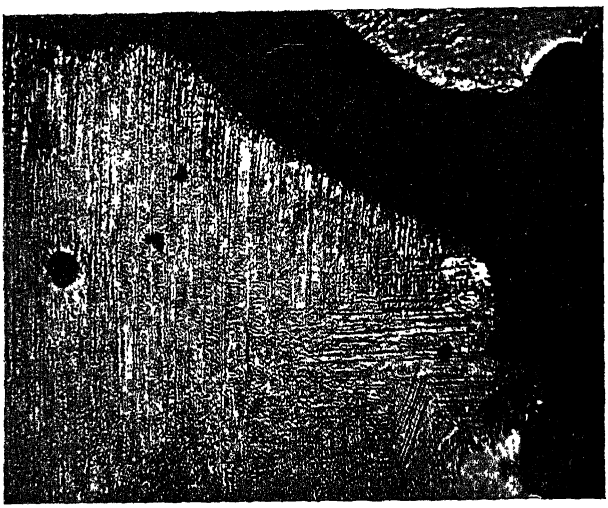

FIG. 23.—MECHANICAL TWINS IN MANGANESE STEEL.

If the twins are represented by heavy lines, and the lines of translation by fine ones, the scheme, Fig. 22, is obtained. The octahedral faces are, at the same time, planes of translation, planes of twinning, and planes of junction. The occurrence is frequent in the cubic system.

The same twins are obtained in manganese steel quenched at yellow heat, but they are localized around the lines of fracture, and are due to the tensile stresses which have caused these fractures (Fig. 23, of 200 diameters, and Fig. 24, of 800 diameters, etched with 5-per cent. alcoholic picric acid).

When the same metal is subjected to an alternative series of polishings and etchings, reliefs form around the patches of

cementite or other foreign substances which have not been dissolved, and on these reliefs the polishing alone results in the formation of twinned lamellæ (Fig. 25; 200 diameters; etched

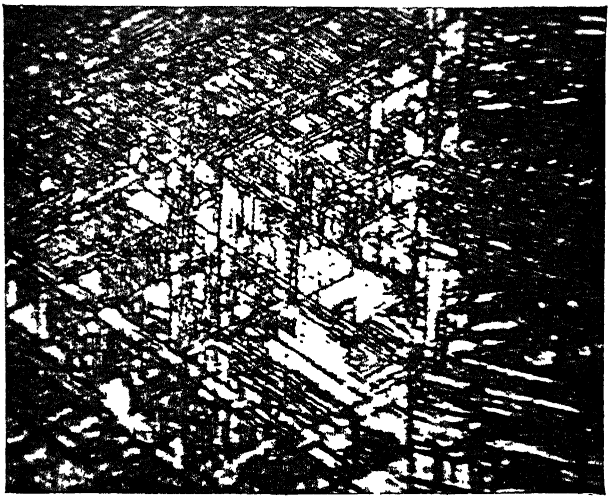

FIG. 24.—MECHANICAL TWINS IN MANGANESE STEEL.

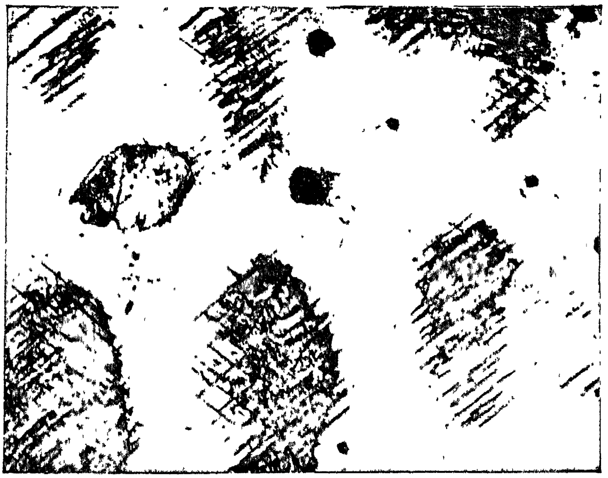

FIG. 25.—TWINNED LAMELLÆ IN MANGANESE STEEL.

30 seconds by a solution of ferric chloride, containing, per cent., 10 parts of concentrated chloride and 6 of hydrochloric acid).

Figures 23, 24, and 25 recall martensite exactly, and thus offer an easy explanation of its structure: the partial transfor-

mation of γ-iron into α-iron, a transformation which starts below 400° C. in the case of sudden quenching, and causes considerable stresses. These stresses, in their turn, involve a more or less complete formation of an infinity of twins, parallel in each grain to the four pairs of octahedral faces; hence the frequency of square figures and equilateral triangles on a chance section.

The structure of martensite is hence a structure peculiar to γ-iron, although the iron is not present in the γ-state, at least for the most part. Even after tempering, when all the iron has resumed the α-state, if the temperature and duration have been sufficiently controlled to prevent the reconstitution of equiaxial grains which characterize α-iron, the α-iron may be retained pseudomorphic on the martensite structure of γ-iron. The grains are in this way cut up by an infinite number of extremely thin lamellæ parallel to four different planes; the continuity of the cleavages $p(001)$ is broken, and the natural fragility of α-iron, due to these cleavages, is evaded. Hence the part played by quenching and tempering in the amelioration of mild steels.

This structure of martensite is that of octahedral meteoric irons on a reduced scale. It is known that these irons are formed of comparatively thick lamellæ parallel to the four pairs of regular octahedral faces. Disregarding the tænite (alloy rich in nickel), the schreibersite, $(Fe, Ni)_3P$, and the plessite (mixture of tænite and kamacite), which may be found interspersed among the lamellæ, the latter are composed of α-iron containing about 7 per cent. of nickel in solid solution, and have received the petrographic name of kamacite (from καμαξ, beam). This is still a structure of γ-iron. Although the iron has resumed the α-state, this structure is preserved because, in the presence of nickel, the point of transformation A 3 is lowered to such an extent that the α-iron cannot resume its natural structure of equiaxial grains. The position is exactly that of martensite tempered at a moderate temperature. The α-iron remains crystallized on the axes of the γ-iron.

Linck was already convinced that he could affirm octahedral meteoric irons being polysynthetic aggregates of four twinned cubic sub-elements with a dominant cube, following the ordinary law: $a^1(111)$ plane of twinning and plane of junction. We

have tried to verify this affirmation on a fragment of a meteorite brought from the neighborhood of Timbuctoo by Mr. Ward. This fragment, taken from near the periphery, was unfortunately a little deformed. For this reason we have not been able to put Linck's law to the test by the study of Neumann's lines, on a section parallel to a face of the dominant cube. The directions observed differed too much from the theoretical to allow a verification of the law; but this difference would be explained by the notable deformations manifestly sustained, the least deformation causing the angles to vary rapidly. In fact, the method was too delicate in character. We then turned to a coarser and consequently more appropriate method, that of pressure-figures. If Linck's theory is true, a face $p(001)$ of the dominant cube cuts the associated four twinned secondary cubes on the planes $a\frac{1}{2}(122)$. Every pressure-figure on such a face of the dominant cube ought then to be characteristic either of a plane (001) or of four planes (122) variously oriented, and no other figures ought to be formed. This is what experiment has fully confirmed, verifying again, in this instance, the conclusions to which Linck had been led by other methods.

Let us add that this polysynthetic structure is by no means peculiar to iron. There is a tendency for it to take place whenever allotropic or isomeric changes are produced in the solid state with a change in volume, so that the resulting tensions can effect mechanical twinning. It is in this manner that Mr. Breuil has been able to show martensitic structure in hardened aluminum bronze,[18] a fact which has been confirmed by Dr. Guillet.[19] These are only specific cases of a general phenomenon.

2. *Congenital Twinning.*

This can only be encountered in γ-iron, which alone crystallizes from the liquid state. But as we have said, in connection with octrahedral meteoric irons, these twins of γ-iron could, under favorable circumstances, be preserved in the α-state. It is in this way that Tschermak tried to explain Neumann's lamellæ. But this explanation, although it did not prove reliable in this particular case, is plausible in itself; and it may

[18] *Comptes Rendus*, vol. cxl., p. 587 (1905).

[19] *Revue de Métallurgie*, Mem., vol. ii., pp. 567 to 588 (1905).

yet be asked if adjacent grains of a crude ingot of iron, each of which grains can represent a primitive crystallite or part of one, could not form twinned groups among themselves.

It is this that we have endeavored to investigate on the same crystal of iron of which we have already spoken, and which has served us in studying the crystallographical position of Neumann's lamellæ. The rectangular parallelopiped cut on three cleavage-faces contained, in fact, besides the dominant crystal, fragments of three or four adjacent grains. If a face $p(001)$ is etched by Heyn's reagent (12-per cent. double chloride

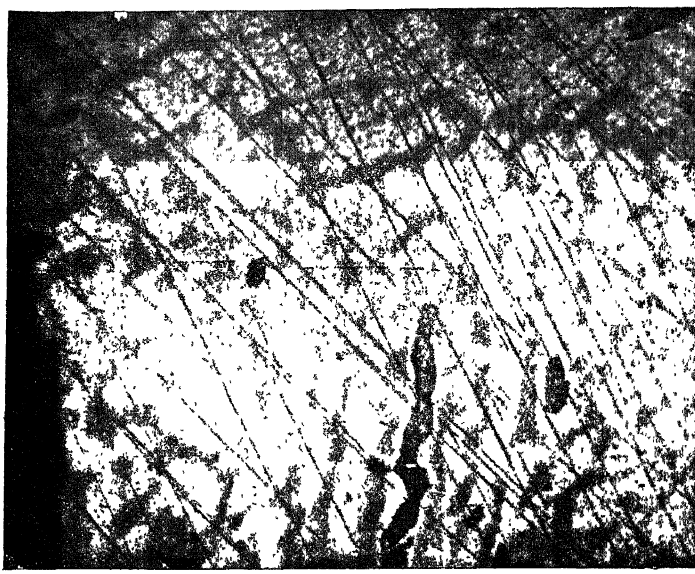

FIG. 26.—DOMINANT AND FOREIGN CRYSTALS.

of copper and ammonium) during 30 seconds, then by (1 to 5) nitric acid, to eat out Neumann's lines, the sections of the foreign grains viewed in vertical light have in general a much deeper tint than that of the faces $p(001)$ of the dominant cube. Moreover, on the latter themselves darker veinings can be seen. Fig. 26 (10 diameters), the sides of the photomicrograph are parallel with the sides of the face of the cube. When these veinings or marblings are studied it is seen that, as regards direction, they are related to the foreign grains. The micrograph reproduced furnishes an example: it shows seven dark bands slightly inclined to the vertical; the middle and longest

one is more pronounced than the others, and is not crossed by the Neumann's lines which meet it; this is a fragment of a foreign crystal, whereas the parallel bands crossed by Neu-

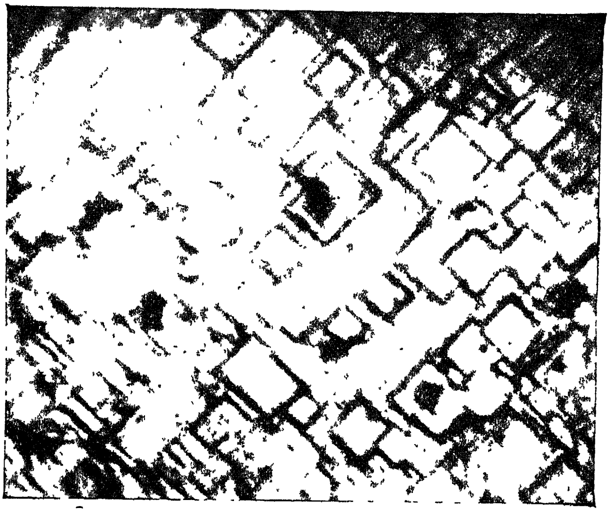

FIG. 27.—CORROSION FIGURES.

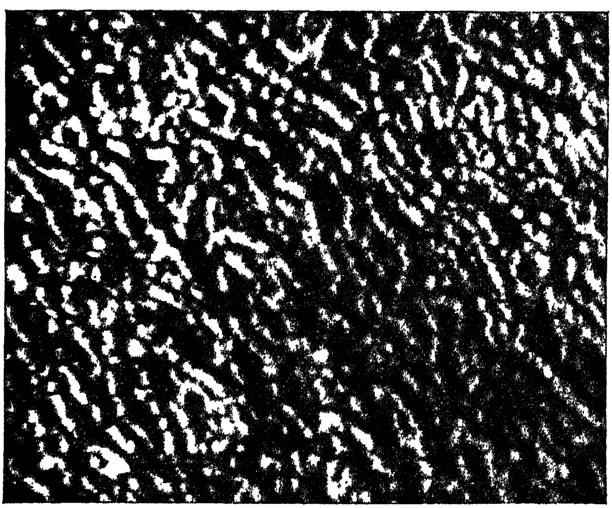

FIG. 28.—CORROSION FIGURES.

mann's lines form integral parts of the dominant crystal. Under high magnifications of the microscope, while the plain portions show beautiful square corrosion figures (Fig. 27, 1200

diameters, with the sides of the photomicrograph parallel to the diagonals of the square), the dark marblings show much smaller and less decided figures, separated by furrows strongly depressed, where, with the light and focus delicately adjusted, square forms can be recognized (Fig. 28; 1200 diameters). There is no doubt that these marblings represent crystallite branches which arise from one of the neighboring grains after solidification. These branches in course of a very slow cooling were assimilated below the point *A*3 by the crystal which they penetrate. Only, although the assimilation was crystallographically complete, since Neumann's lamellæ have nothing to do with the marblings, there yet remained some recognizable traces of the difference of origin, which are rendered evident by the copper chloride.

The grains that retained their own orientation different from that of the dominant crystal, are the residues of the primary crystallites. Therefore, from the actual orientation of the foreign grains we can see if these primary crystallites are formed in the position of twins. In that case, the face of the dominant cube will cut the twinned cubes on the planes $a\frac{1}{2}$(122), as happens in the octahedral meteoric irons. The experiment made with pressure figures only confirmed this conjecture for one grain in three. But the two other grains may originate from a different group; it would therefore be imprudent to draw a definite conclusion either one way or the other.

3. *Twinning Resulting from Annealing after Deformation.*

Such twins as these are produced with greatest ease in copper, bronze, brass, etc., as shown by the work of Prof. Heyn,[20] Mr. Charpy,[21] and others.

When a crystal of iron which has suffered partial deformation—by the pressure of a needle on a face of the cube, for example—is annealed either below *A*2 or between *A*2 and *A*3, the region of most deformation—that is to say, in the case under consideration, a ring round the point of impact—resolves itself into little grains of various crystalline orientation (Fig. 29; 60 diameters; etched with 5-per cent. alcoholic picric acid); but these new grains never show twinned lamellæ.

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[20] *Zeitschrift des Vereines deutscher Ingenieure*, vol. xliv., pp. 433 to 441, 503 to 509 (1900).

[21] *Bulletin de la Société d'Encouragement* (5), vol. i., p. 180 (1896).

With γ-iron it is not the same. Our sample of nickel-chromium steel, which, as rough cast, contained no twin, developed a large number when cold-deformed and annealed at about 1,300°. It remained to determine these twins, although from our knowledge of meteoric irons it, *a priori*, was highly probable that we again had to do with a^1 twinning.

The grains were much too small, with a mean diameter of 1 mm., to permit of isolation and slicing, so a chance cut was made. It was probable that on this section, which contained a hundred or so grains, there would be some presenting spontaneously a cube face. To find them, the whole

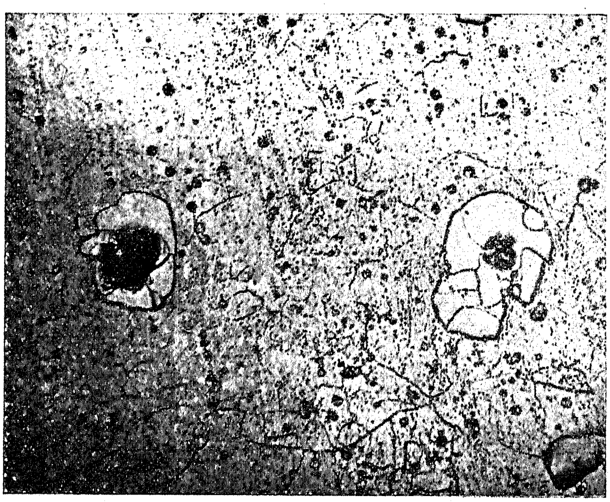

FIG. 29.—IRON ANNEALED AFTER DEFORMATION.

piece was subjected to slight general deformation in two directions at right angles, so as to show lines of translation in the polished section, and microscopic search was made for grains presenting two systems of rectangular lines and no others. We found one of sufficient size fulfilling these conditions (Fig. 30, above the line $A\ B$). This grain is cut precisely by lamellæ $A\ a\ q\ b$, $c\ d\ e\ f$, $g\ h\ o\ B$, $q\ p\ r\ s$, parallel to one of the systems of lines, without having the second: they are twins, and we happen to be on a face $p(001)$ of the dominant cube, or nearly so. To make certain, indentings were made with the needle, and the cross of the cubic faces was obtained in good form, and on one of the twins a different figure. However, on

looking closely into the matter, it was observed that the arms of the cross inclined slightly to the vertical lines; that the crosses gave rise to a new system of lines also slightly inclined to the vertical, and that the line $a\,b$, boundary of the twin that we have figured perpendicular at $A\,B$, is not exactly so. This was therefore not a true $p(001)$ face, but it did not deviate much from one. To rectify matters, a cut was made perpen-

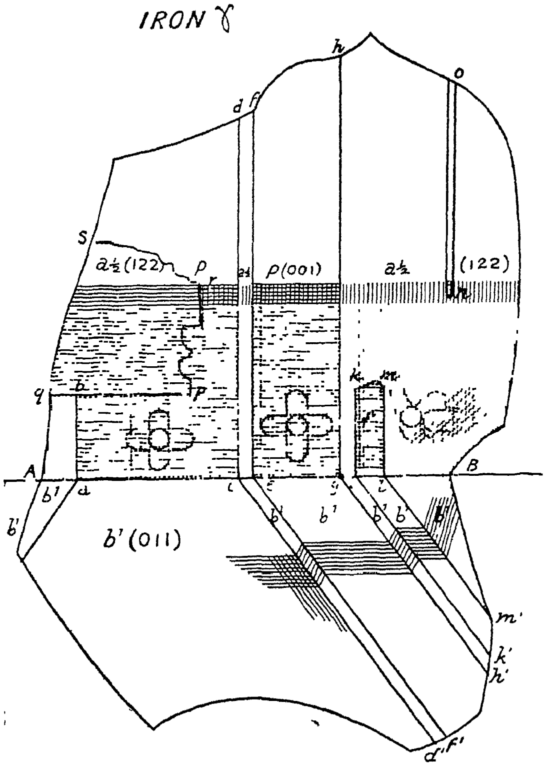

FIG. 30.—TWINS DEVELOPED BY COLD-DEFORMING AND ANNEALING.

dicular to the first following $A\,B$, and this is represented by the lower part of Fig. 30, after reduction to the plane of the picture. This section encounters the twins $A\,a\,b'$, $c\,e\,d'\,f'$, $g\,i\,h'\,k'$, $l\,m'\,B$, belonging to two different systems.

If it is a true $b^1(011)$ face, and if our conjectures concerning the nature of the twinning are well founded, the lines $a\,b'$, $c\,d'$, etc., should make an angle of 54° 44′ with $A\,B$. In reality, as may be supposed after the remarks made, these conditions were not well fulfilled. But taking the direction of the lamellæ

for a guide, the first cut was adjusted on a fine grindstone, then the second, and so on progressively. When the work was done, the two rectangular faces ought to be exactly $p(001)$ and $b^1(011)$ respectively; and the lamellæ ought to make with AB an angle of 90° on the first and of 54° 44′ on the second. From measurements made on the photographs with a protractor, the angles are respectively 89° and $54\frac{1}{2}$°, which is a satisfactory result, more particularly as the production of lines of translation necessitated a slight deformation. The twinning resulting from annealing γ-iron is then the same as the mechanical twinning, with $a^1(111)$ for plane of twinning and plane of junction; it is possible, as we have already said, that the annealing simply develops the germs of mechanical twins.

4. *Mechanical Properties Functional of Crystalline Orientation.*

α-Iron.—Two prismatic test-pieces were taken in the same crystal previously annealed at about 800°, and cut in such a way that one of the axes of the figure was in the one case parallel to a quaternary axis of the crystal (lateral faces parallel to $p(001)$, $p(001)$, $b^1(011)$, $b^2(012)$); in the other case to a ternary axis (lateral faces parallel to $b^1(011)$ and $a^2(112)$).

These two test-pieces were submitted to compression, with the following results:

	Compression Parallel to	
	Quarternary Axis.	Ternary Axis.
Original height of prism in millimeters	11.22	8.905
Original section of prism in square millimeters	96.40	79.94
Limit of elasticity in kilograms per sq. millimeter	13.9	17.0
Maximum charge in kilograms per square millimeter of original section	33.5	38.8
Final height in millimeters	10.48	8.34
Crushing, total, in millimeters	0.74	0.565
Crushing, per cent. per kilogram above the elastic limit	0.34	0.29

Trials of hardness were also made by Brinell's method, under a charge of 140 kg., applied during 2 min. with a ball of 5 mm. diameter, with the following results:

Metal Annealed at Bright Cherry Red (about 800°).

	$p(001)$.	$b^1(011)$.	$a^1(111)$.
Diameter (D) in millimeters	1.642	1.602	1.533
$\pi D^2 : 4$ in square millimeters	2.1174	2.0157	1.8457
Hardness (H)	66	69	76

Metal Annealed at Very Dull Red (about 550°).

	$p(001)$.	$b^1(011)$	$a^1(111)$.
Diameter (D) in millimeters	1.540	1.500	1.484
$\pi D^2 : 4$ in square millimeters	1.8627	1.7672	1.7296
Hardness (H)	75	79	81

The number for the mean hardness of metal annealed at bright cherry red (70) is notably lower than the minimum (76) of Dr. Benedicks[22] for iron also annealed; this is explained by the absence of joints in our sample.

Each of our numbers is the mean of four impressions measured in two rectangular diameters. These impressions are neither so sharp nor so circular as on metals with fine grain, where they affect a large number of grains of varied orientation. Consequently there is some uncertainty in the measurement of the diameters, and as the differences appropriated to different faces do not much exceed the limit of experimental errors, the question may be asked if these differences are quite positive. We do not think, however, that doubt can be cast on them, since we have two simultaneous series concordant among themselves, with the compression tests and with the general law which assigns the minimum hardness to cleavage-faces.

The conclusions are that the mechanical properties of α-iron vary with the crystallographic orientation.

γ-Iron.—For this purpose Mr. Hadfield's manganese steel was used. The grains were sufficiently large to permit of test being made with the ball.

[22] Thesis, Upsala, 1904.

On face $p(001)$ the impressions are not circular: the diameters parallel to the diagonals are about 10 per cent. longer than the diameters parallel to the sides, by reason of swelling in the direction of the former. Means were taken.

On faces b^1 and a^1 the impressions are circular.

Under a charge of 200 kg., applied during 2 min. with a ball 5 mm. in diameter, the results were:

	$p(001)$.	$b^1(011)$.	$a^1(111)$.
Number of impressions	3	4	2
Diameter (D) in millimeters	1.147	1.145	1.150
$\pi D^2 : 4$ in square millimeters	1.033	1.030	1.039
Hardness (H)	193	194	192

These numbers indicate a very moderate maximum of hardness on face $b^1(011)$, but the differences between different faces are less than those observed between two adjacent impressions on the same face. γ-iron is hence practically isotropic as far as the mechanical properties are concerned.

5. *Corrosion Figures.*

The corrosion figures of α-iron on face $p(001)$ have been described by Mr. Stead and Mr. Heyn. It is known that these are squares parallel to the edges of the cube face. We have given above a photograph (Fig. 27). On β-iron we have obtained corrosion or growth figures spontaneously in our previous tests on the crystallization of iron; they are the same as for α-iron.

On γ-iron, still working on face $p(001)$, the lines of corrosion are again parallel to the sides of the square, only they are neither so continuous nor so soft as on α-iron. As reagent, the double chloride of copper and ammonium, or 10 per cent. nitric acid, can be used for the manganese steel, while for nickel-chromium steel, iron perchloride, acidified with hydrochloric acid, is preferable: the etching is irregular, and, even after deformation, followed by annealing at 1,300°, shows the primary crystals very clearly. It is well to polish it lightly

again. The appearance Fig. 31 is then obtained, which corresponds to a magnification of about 200 diameters. The drawing shows a face $p(001)$ and an adjacent face $b^1(011)$, with a twinned lamella.

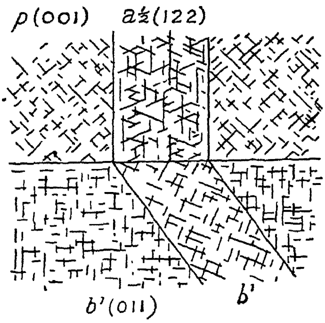

FIG. 31.—IRON CORROSION FIGURES.

To put it briefly, corrosion figures, up to the present, have not furnished useful distinctive characters.

6. *Synchronous Crystallization Figures.*

We heated to redness a crystal of iron having a cube face polished in a nickel crucible, beneath a layer of magnesia cal-

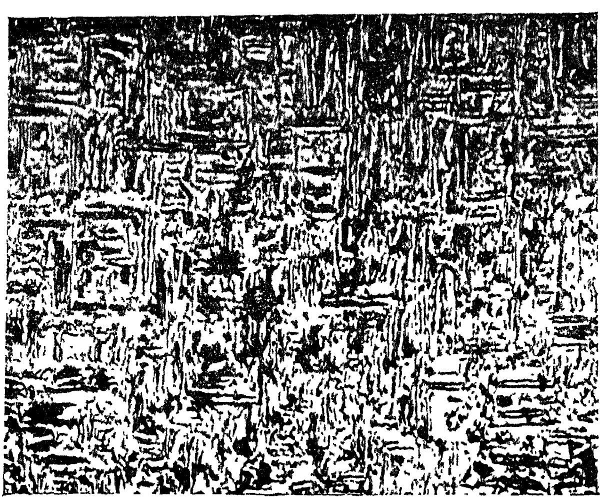

FIG. 32.—CRYSTALLIZATION FIGURES.

cined at a high temperature, which in its turn was covered with a layer of cast-iron shavings. The polished face oxidized naturally, and displayed square figures strongly resembling corro-

sion figures (Fig. 32; 400 diameters), the sides of the photomicrograph being parallel to the edges of the cube face. It seems that the oxidation has been regulated by the structure of

FIG. 33.—BORIC ACID MARKINGS.

FIG. 34.—BORIC ACID MARKINGS.

the metal. We have tried to repeat the experiment under known conditions of temperature, but without success.

In another test, we introduced a crystal of iron, again having

a polished cube face, into a bath of molten boric acid, at about 800° C., the temperature being gauged by the eye. Needles of boric acid congealed at first on the polished surface, and preserved it provisionally from attack in such a way that the silhouette of the needles was imprinted on the iron (Fig. 33; 200 diameters). The interesting point is that the needles of boric acid, instead of arranging themselves on their own account in their natural order, as in Fig. 33, may follow the direction of the system of the iron; the appearance Fig. 34 (400 diameters) is then obtained, of which the sides are parallel to the edges of the cube face. In this figure some directions parallel to these edges may be seen, and others which seem to be parallel to the directions of Neumann's lines. These directions are really somewhat uncertain, because the needles of boric acid form fan-shaped groups, and, what is more, the experiment does not always take place in the way desired. Researches in this direction might, perhaps, merit resumption and following up.

7. *Segregation Figures.*

Crude cast manganese steel in the vicinity of the pipe presents details of structure that demand studying on their own account. Little hard nuclei are encountered, to which other constituents are attached, and notably lamellæ of cementite, crystallographically oriented.

On a cube face the nuclei occupy the summits of a square system. Fig. 35 shows the result of simple polishing at 65 diameters. The lamellæ which start from the nuclei, and others which are isolated in the metal, are parallel either to the edges of the cube face or to its diagonals, or to the lines which join the summits to the centers of the opposed edges. These lamellæ are too small to permit of their being followed on two adjacent cube faces: hence it is impossible, at least with the sample at our disposal, to determine exactly all the planes of segregation. Only it may be remarked that the lamellæ are of two kinds, and of decidedly different scales of magnitude, and that the larger ones are those which follow the diagonals of the square. It is concluded from this that the principal planes of segregation of γ-iron are again the planes $a^1(111)$; but that secondary planes also may exist $p(001)$, $b^1(011)$, $b^2(012)$, $a^2(112)$, or $a^{\frac{1}{2}}(122)$, among which we should not know now which

to choose. It is the lamellæ of cementite which contribute the relative brittleness to crude cast manganese steel, and cause it to break. The fracture then shows brilliant little facets disposed in cups; these are really the lamellæ. The quenching

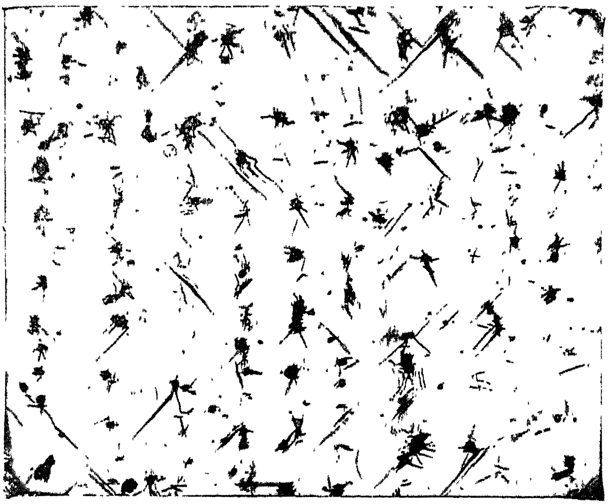

FIG. 35.—SEGREGATION FIGURES IN MANGANESE STEEL.

keeps the cementite in solid solution, and so communicates all the characters which distinguish the metal. In the same way, if hypereutectoid carbon steels are cooled from about 1,100° quickly enough, without actual hardening, the cementite is distributed in the planes of segregation of γ-iron.

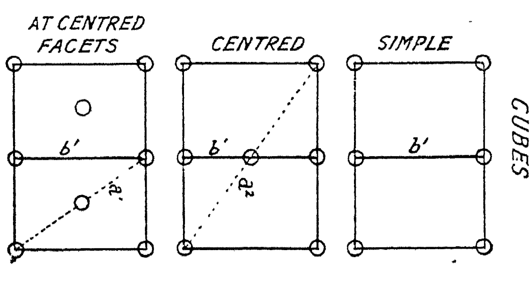

FIG. 36.—INTERSECTIONS OF CUBES.

Rose established that, in the cubic meteoric irons of Braunau and Seeläsgen, the rhabdite, $(Fe, Ni)_3 P$, is oriented in accordance with the three cube faces. Tschermak confirms this fact, and adds another possible plane of segregation parallel to

Neumann's lamellæ—that is to say $a^2(112)$. According to Kunz and Weinschenk, the rhabdite in the Floyd Mountain meteorite is parallel to Neumann's lamellæ. (All these notes relating to meteorites are borrowed from the frequently cited book of Cohen.)

Inasmuch as phosphorus tends to the elimination of point $A3$, it is possible that the planes of segregation $p(001)$ and $a^2(112)$ of rhabdite characterize either β- or α-iron. Systematic experiments on the crystallographic segregation of phosphorus, at known temperatures and on samples of known composition, would therefore probably be interesting. Mr. Stead, who has made such brilliant studies of the relations between iron and phosphorus, would be better fitted than any one else to conduct them successfully.

III. Summary.

The results of this research are set forth in the following table:

	α	β	γ
Planes of translation	$a^1(111)$ difficult	none known	$a^1(111)$ easy
Folds	dominant	exclusive	absent
Mechanical twinning: planes of twinning	$a^1(111)$	none known	$a^1(111)$
Mechanical twinning: planes of junction	$a^2(112)$		$a^1(111)$
Twinning by annealing: planes of twinning	none known	none known	$a^1(111)$
Twinning by annealing: planes of junction			$a^1(111)$
Face of maximum hardness	$a^1(111)$	?	$b^1(011)$?
Planes of easiest etching	$p(001)$	$p(001)$	$p(001)$

It is very difficult to interpret these results. Two courses suggest themselves.

Firstly, it may be supposed, with some probability, that the planes of translation, of twinning, of cleavage, and of segregation, are planes of the greatest reticular density, something like the walls and floors of a house. Moreover, it is known that there are three varieties of the cubic system: the simple cube, which has intersections only at its summits; the centered cube, which has another intersection at its center; and the cube with centered faces, with an intersection at the center of each of its faces. (These three cubes are represented by Fig. 36, projected on one of their diametrical planes; the little cir-

cles indicate the intersections.) In the simple cube the plane of the greatest reticular density is the plane $p(001)$, and among the truncations on the angles, the plane $a^1(111)$. In the centered cube the number of intersections is doubled on $b^1(011)$ and $a^2(112)$, and is not modified on $p(001)$ and $a^1(111)$, so that the reticular density becomes greater on b^1 than on p, greater on a^2 than on a^1. In the cube with centered faces the number of intersections is doubled on p and b^1, quadrupled on a^1, which becomes the plane of greatest reticular density. If it is now observed that p is a plane of perfect cleavage and minimum hardness for α-iron, and that a^1 plays the part, by far the most prominent part, in the crystallography of γ-iron, one is led to think that if the mesh of α-iron is a simple cube, and that of γ-iron a cube with centered faces, then that of β-iron would be a centered cube. As, on the other hand, the number of intersections is equal to that of the cube meshes for the simple cube, double for the centered cube, and quadruple for the cube with centered faces, each allotropic transformation of iron will be characterized by a division into halves of the molecular polyhedron with rising temperature. This is a very simple view.

However, the plane $a^2(112)$ has a greater reticular density than $a^1(111)$ in the centered cube, and not in the simple cube, hence the Neumann lamellæ, which demand $a^2(112)$ as plane of translation, could not be attributed to α-iron. And one is led to attribute them to β-iron, which should be formed temporarily under the influence of shock. This is no unreasonable supposition, and may also be arrived at by other considerations, notably by the experiments of Curie and of Morris on the laws of the appearance and disappearance of magnetism.[23]

Secondly, Mr. Wallerant considers the mechanical twinning as a proof of merosymmetry.[24] That is another course to follow.

Really these attempts at interpretation are probably premature in the present state of the crystallography, but they can be used as working hypotheses. The only positive conclusion that we can draw from these researches is, that the three allotropic varieties of iron, although they all crystallize in the

[23] *Metallographist*, vol. ii., pp. 169 to 186 (1899).

[24] *Bulletin de la Société française de Minéralogie*, July, 1904.

cubic system, present well-marked specific characters, and cannot have the same internal structure.

IV. Appendix.

Nickel.—Pressure-figures were made at a temperature above the disappearance of magnetism, on a polished surface of a sample of nickel, containing a little iron and manganese, and previously cold-deformed and annealed at a white-heat. These indentions may be made in the open air, nickel being much less prone to oxidation than iron. Around each point of impact, lines of translation—that is to say, straight ones—were obtained.

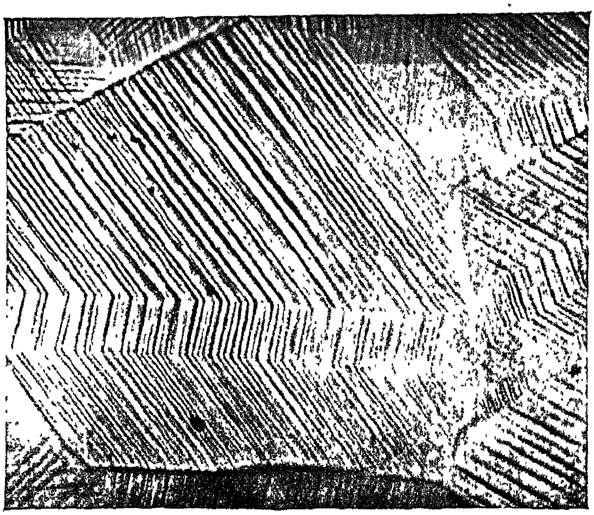

Fig. 37.—Pressure-Figures on Nickel.

Then, under the microscope, a new indention was made in the cold alongside each of the indentions made at the high temperature upon any grain with sufficient surface. In every case, the lines produced in the cold are also straight, and just like those produced on the same grain while hot. α-nickel and β-nickel have the same plane of translation, $a^1(111)$, which is also plane of twinning by annealing for β-nickel. It is true, as Professor Le Chatelier has said, that β-nickel corresponds to γ-iron; α-nickel corresponds to α-iron, with this difference, that it does not give curved lines of deformation.

We have also made deformation figures on the iron-nickel

alloy called "invar," of which the coefficient of dilatation is almost nil at the ordinary temperature, and which is in consequence in course of transformation at this temperature. It might be asked if this circumstance would not have some influence on the deformation lines. It has not any. Fig. 37 represents a section of this metal after slight deformation and previous polishing at 200 diameters. This section shows beautiful lines of translation. The twins seen were pre-existing. The deformation has only put them in evidence.

Improvements in Rolling Iron and Steel.

BY JAMES E. YORK, NEW YORK, N. Y.

(London Meeting, July, 1906.)

THE honor so fairly earned and so incompletely and tardily paid to Henry Cort, the inventor of the puddling-furnace and the rolling-mill, has been fully set forth by Mr. Charles H. Morgan,[1] and needs no further emphasis here.

In view of the importance of the rolling-mill in the treatment of iron and steel, the paucity of information concerning it is surprising. With the exception of a book, written about 35 years ago, by Peter Ritter von Tunner, an Honorary Member of this Institute, and of desultory articles on simple sections, which have appeared in technical journals, I know of no publication attempting to treat this subject, although volumes have been written on the scientific theory and technical manipulations involved in other branches of the iron- and steel-industry. This strange anomaly may be due to the fact that roll-turning cannot be called a scientific business. It does not ordinarily come within the range of an educated engineer; yet it cannot be performed by an ordinary mechanic. It demands some of the qualifications of an engineer, in designing, and those of a mechanic, in execution. The men who have followed this trade have controlled the training of their successors. In many cases the technical knowledge required has

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[1] The Case of Henry Cort, *Trans.*, xxxv., 893 to 902 (1905).

been handed down from father to son, and there has been a motive of private interest to prevent its public dissemination. Moreover, the statement of the various principles involved in the process would require, besides manual experience, a degree of scientific knowledge which roll-turners do not usually possess; and, finally, any book on the subject, in order to be of real, practical value, would have to be so fully illustrated as to make it very costly to publish.

Regarding my own experience, I may say that I served a six-year apprenticeship at the trade of roll-turning at Wednesbury, South Staffordshire, England, and, later, accepted a position as head roll-turner, believing that I fully understood the technical points of the business; but, after a short time, I realized that, if there were any theories governing the practice, I still had to acquire them.

Forty years ago, the bulk of the metal rolled was iron. Rolling iron was in some regards simpler, and in other regards more difficult, than rolling steel. Iron was adapted to quicker reduction, being softer, and capable of sustaining greater heat without injury. It was inherently weaker than steel at a rolling-temperature, even when properly heated; but piles could be made, conforming to the finished section, which was a great advantage in making flange-sections.

In the early stages of this business, the market called for simple forms only, such as flats, rounds and squares; but with the advent of railroads, and other commercial demands, it became necessary to roll more complex sections, such as rails, beams, channels and tees, which added very materially to the difficulties in the art of roll-designing. These difficulties I will attempt to explain.

Fig. 1 illustrates the rolling of a flat 6 in. wide and 0.5 in. thick. This was undoubtedly the first rolling ever done in grooved rolls, and represents the simplest section that we have to make, and the only shape, rolled in closed grooves, to which the principle of uniform flow of metal from rolling-contact can be applied.

It is recognized by all experts that metal flows under pressure in the direction of least resistance, which, in rolling, is at right angles to the journal of the roll, or lengthwise of the bar. The result is to make the bar stronger longitudinally than

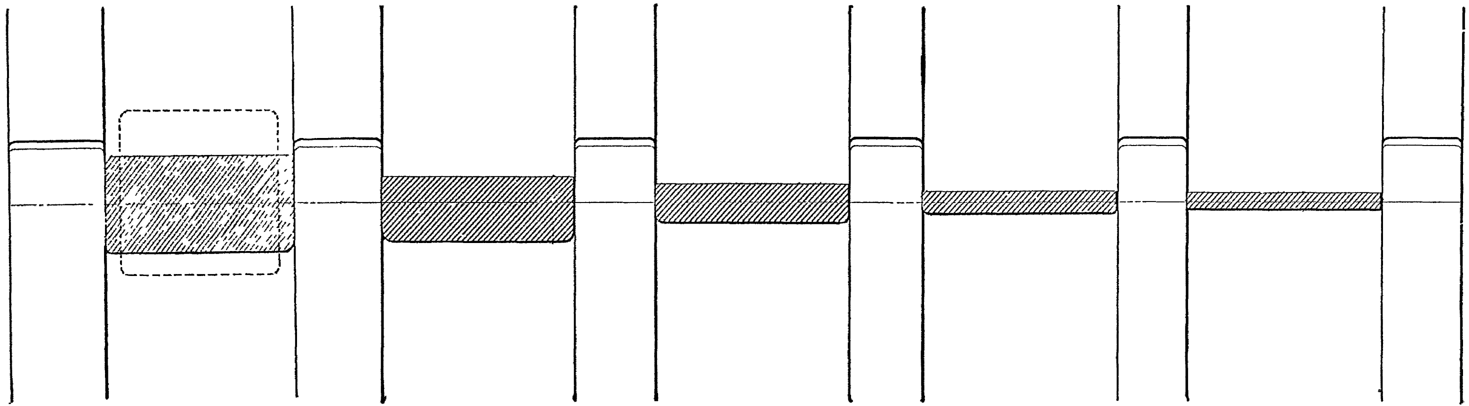

Fig. 1.—Showing Method of Rolling a 6-In. Flat.

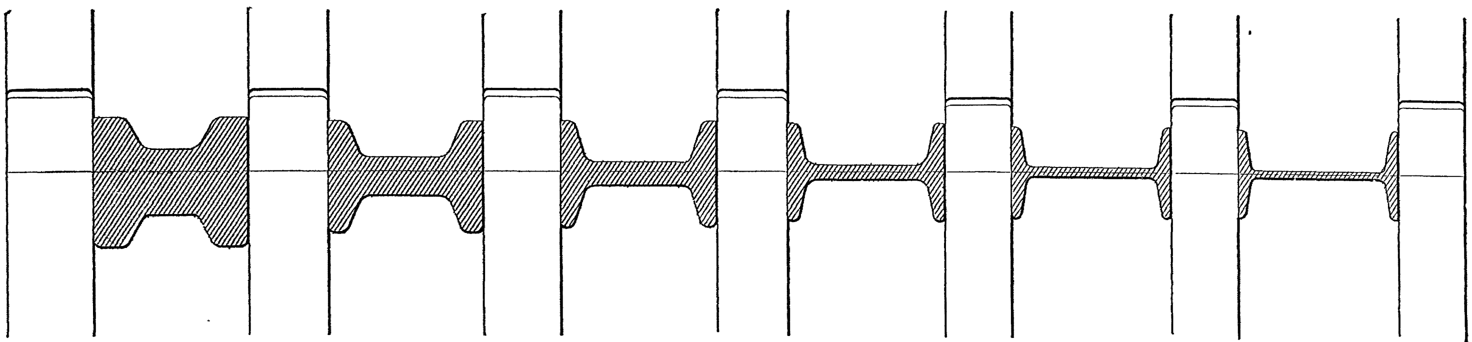

Fig. 2.—Showing Method of Rolling a 6-In. Beam.

transversely; and this is true of every bar (except rounds and squares), either of iron or steel, rolled in grooved rolls. In some cases, the difference amounts to from 10 to 15 per cent. of the strength.

The most important physical effects are produced on the metal during the last stages of rolling. As the temperature decreases and the section becomes thinner, greater resistance is offered to reduction, and there is, consequently, a greater "fining" of the grain in steel or the fiber in iron. It is well known that in large ingots there is scarcely any "fining" of the grain by rolling, and that this effect is reached only when comparatively small dimensions have been attained. (Of course, the result is influenced also by overheating, and by the chemical composition of the metal.)

The only important modern change in the process of rolling is the addition of a third roll, which doubles the capacity of the mill, and constitutes the "three-high system."

The billet or ingot, from which any flanged section is rolled in an ordinary mill, must have dimensions practically equaling or exceeding the extreme dimensions of the section to be produced, although the actual area of metal in this section will be much smaller than that of the original mass. Moreover, the first shaping-groove must be wider than the mass, in order that the latter may enter it. It follows that the reduction of the metal by rolling-pressure is wholly in one direction; and this result is inseparable from the conditions presented by ordinary mills.

Fig. 2 shows how we rolled, "two-high," an ordinary 6-in. beam. Modern mills are "three-high;" but the same principle governs. This is one of the easiest flange-sections we have to roll, because it has an equal amount of metal on each side of the pitch-line, or center of the mill, and the flanges are of equal dimensions.

The difficulty encountered in rolling a section like this (and found to be still greater in sections of unsymmetrical form) is involved in the displacement of the metal from a square or flat billet, to form a web. This displaced metal runs to length; and since there is no commensurate reduction of the flange-parts of the section, the metal of those parts is liable to crack, because it is not rolled, but stretched. After the blank in the

first pass has been filled out, a more uniform reduction of the various parts can be provided for. The finished section shown in Fig. 2 has a flange 3.33 in. wide, with a web about 0.23 in. wide—the latter having received a transverse reduction about twelve times that of the cross-section of the flanges.

The rolling of flange-bars involves the thinning of the flange sideways by forcing into the groove, metal much thicker than the recess provided. This is rather a wedging and drawing than a rolling action; and all such flanges are tapered, to permit their exit from the rolls. At the ends of finished bars of this character considerable waste is created by the different roll-dimensions, which prevent uniform surface-speed of the metal during the operation.

Another objection to this method of making flange-bars is, that it is impossible to distribute the metal in scientific proportions. In most of the beams rolled to-day by the prevailing method, the web is thicker by at least 20 per cent. than it need be, to harmonize with the other dimensions of the bar. The reason (if we take Fig. 2 as an instance) is, that the part of the roll which forms the web is 3 in. greater in diameter than the part which comes into contact with the widest part of the flange, and has at least 9 in. per revolution greater surface-travel than the part in contact with the end of the flange. The flange must, therefore, either slip or stretch, to accommodate itself to the larger diameter. This causes the web to buckle, or corrugate, when rolled down to the proper thickness. These difficulties are greatly increased in rolling wider-flanged sections, whether large or small in other dimensions. In fact, it is almost impossible to produce, in an ordinary rolling-mill, any shape with a flange equal in width to the total height of the section. The effect of the difference in surface-speed of rolling on various parts of rails, beams and other flange-sections, is, in my opinion, the cause of a large number of otherwise unaccountable fractures in service, due, probably, to permanent internal stresses rolled into the bar.

While engaged in the manufacture of iron beams about 30 years ago, it was suggested that, if beams could be rolled with wider flanges, greater height and thinner webs, at least 20 per cent. greater carrying-strength could be secured for a given weight. This led my brother and myself to design and build

what is now known as the York universal mill, for rolling sections with wide flanges and thin webs, or the usual sections when desired.

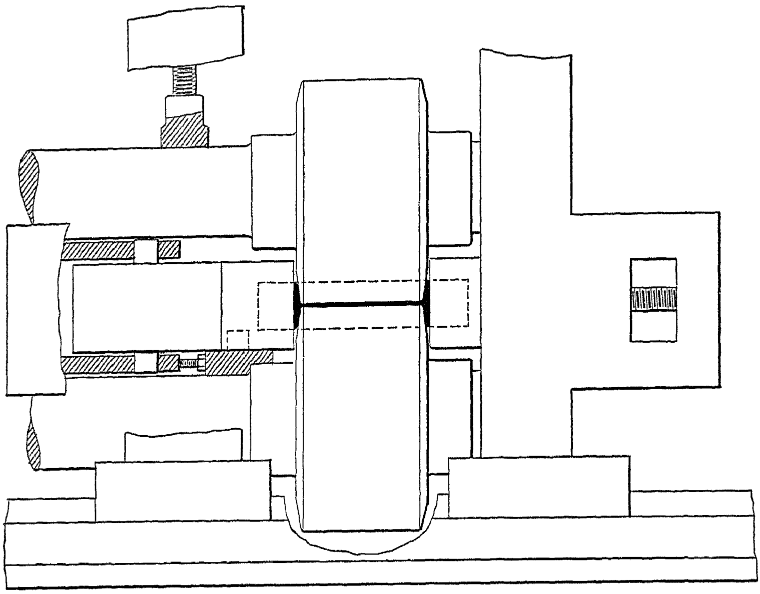

FIG. 3.—YORK UNIVERSAL MILL, SHOWING THE METHOD OF ROLLING AN 18-IN. BEAM WITH 6-IN. FLANGES.

The front elevation and plan of this mill, which is a radical departure from the process of rolling illustrated in Figs. 1 and 2, are shown in Figs. 3 and 4.

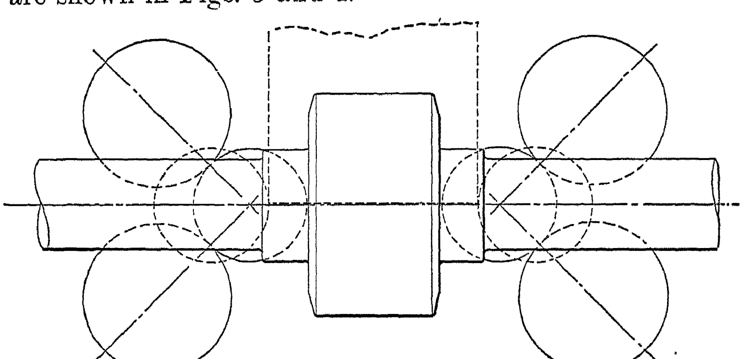

FIG. 4.—YORK UNIVERSAL MILL.

The advantages of this mill are:

1. It dispenses with grooves entirely and gets uniform sections without this provision.

2. It eliminates the wasteful and difficult method of displacing metal to form a web which is out of all proportion to the other parts of the section.

3. No technical knowledge is required to design the rolls.

4. The operation is exceedingly simple, and the same rolls can produce a properly proportioned section weighing 50 or 150 lb. per foot.

5. By the old methods the web increases in the same proportion as the width of flanges, which is a great waste of metal.

6. This mill is adapted to roll either scrap or new steel to sections at least 50 per cent. less in weight per linear foot than the common forms.

7. The beam produced by this mill has the same tensile strength transversely (where it is required) as in the longitudinal direction.

8. The rolls can be adjusted either at the commencement or during the operation, in order to have the web thinner and the flanges thicker, or the reverse.

9. This mill can roll a bar having one flange thicker than the other, since the dimensions of the flanges can be modified to suit requirements.

This mill consists of a pair of horizontal rolls positively driven, and a pair of vertical rolls run by the friction of contact against the bar. A line through the axis of these vertical rolls would intersect the center of the horizontal rolls.

The horizontal rolls work upon the sides of the web and shape the inner contour of the beam, while the vertical rolls shape the outer flange-surfaces. Both horizontal and vertical rolls are connected by a screw-device, so that the movement in all the rolls shall be simultaneous; an arrangement which secures the proper relative reduction of the flanges and the web.

Fig. 3 represents the rolling of an 18-in. beam with 6-in. flanges. The slab, or ingot, for making this section is 29.5 in. wide and 6.25 in. thick. These dimensions give a reduction on each flange of $5\frac{7}{8}$ in. sidewise and a reduction on the web of $5\frac{7}{8}$ in. at the same time, which produces a beam 18 in. by $\frac{3}{8}$ in., having the usual taper to the flanges.

The rolls used in this mill weigh only one-quarter as much as those ordinarily used, and the power required is much less than that of the ordinary mill, for the reason that it is

a rolling-process, both horizontal and vertical, instead of a partly wedging and stretching action, as shown in Fig. 2.

The mill is adapted, without changing rolls, to make any reasonable weight of beam, having wide or narrow flanges up to the width of the face of the vertical rolls. It also rolls tees of different dimensions having any length of leg or flange in the beam-rolls, the vertical rolls forming the flange. A beam-section cut longitudinally through the web forms two tees of half the height of the beam; hence a tee can be rolled on either side of the mill. Tee-sections, about the most difficult to roll in an ordinary mill, are very easily made by my process.

It was my intention, when building this mill on a commercial scale, to have a blooming-mill in which to roll an ingot down to about 3 in. thick in the web and a proportional thickness in the flanges; but, for unavoidable reasons, it became necessary to use the finishing-mill to do the entire reduction from the ingot, or slab, to the finished section. This action produced a slight defect in the finished product, the bars not being exactly parallel in the width of the flanges throughout the entire length. In order to overcome this difficulty, I proposed to use a pair of supplementary rolls to work on the edges of the flanges, these rolls to be placed in line of the main rolls, and to be run at the same surface-speed of delivery as the rolls in the mill proper. These changes have since been installed, and beams of perfect section are now rolled.

Our experience with this process in rolling beams, having the width of the two flanges greater than the height of the web, showed that it was necessary to feed-in the side-rolls faster than the horizontal; otherwise, in rolling light webs with wide flanges, there would be a buckling-action in the web, caused by the greater roll-surface contact on both sides of the flange. The mass being, of course, stiffer in the direction of the cross-section, the web was then held back to the average speed of the flanges, which caused it to corrugate.

The advantage of a light-web beam is illustrated as follows: The 18- by $\frac{3}{8}$-in. beam, Fig. 3, weighs 42 lb. per ft., and the nearest comparable size of the ordinary beam is 15 by 0.41 in., which also weighs 42 lb. per foot. The ordinary beam, however, has a load-carrying capacity of 18 per cent. less than the light-web beam, a difference which is larger in most of our

sections, especially those made to take the place of built-up plate- and angle-girders, having a much greater weight per

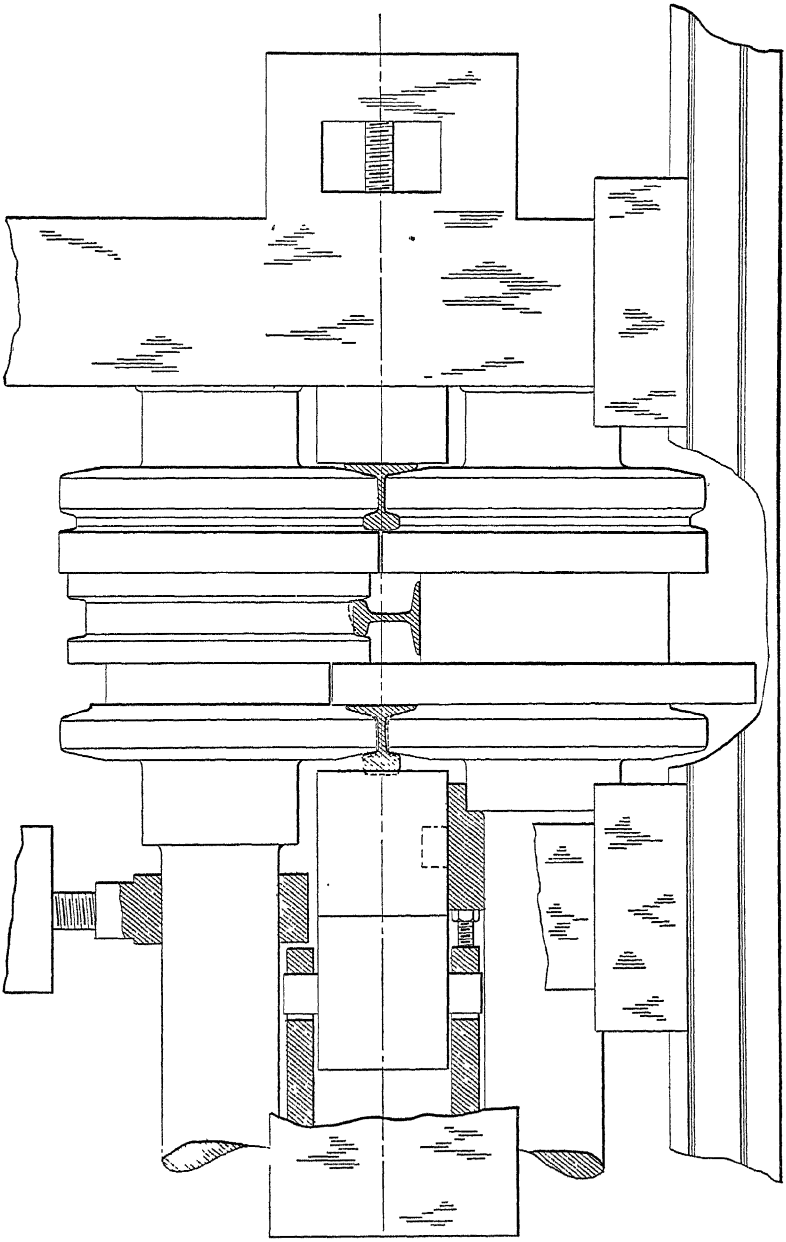

FIG. 5.—SHOWING THE METHOD OF RE-ROLLING A "T"-RAIL BY UNIVERSAL MILL.

foot. In addition to the gain in strength, the cost of building-up is reduced; this cost, in some cases, exceeds 40 per cent. of

the cost of our rolled beams. Sections having wide flanges which equal the height of the section (or nearly so), are intended to take the place of "Z" bars and other columns. In this case, also, there is a great saving in weight and cost.

This mill has also been adapted to re-roll rails of various types for further service in the track, as shown in Figs. 5 and 6. It is admirable for this purpose from the fact that the rolls can be chilled, thus permitting the proper finishing-temperature required to secure the best physical results. This mill can, also, upset metal from the web into the head and leave the

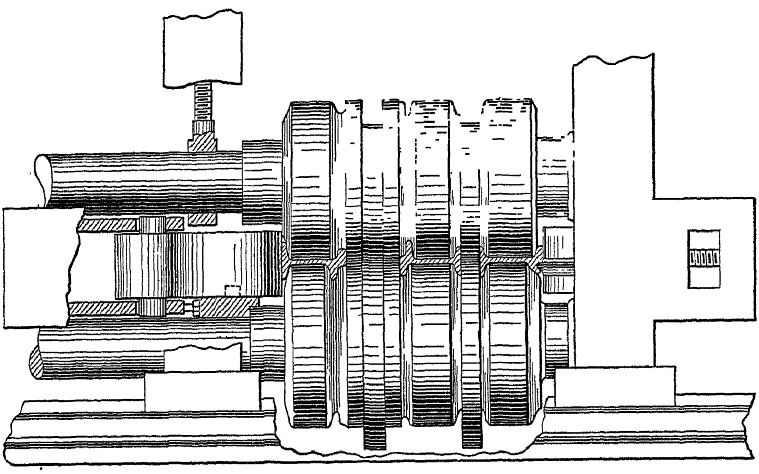

FIG. 6.—SHOWING THE METHOD OF RE-ROLLING A GIRDER-RAIL BY UNIVERSAL MILL.

flange the same width, and it can re-roll any flange-bar or other sections in a longitudinal direction.

Some years ago I took up the problem of making a steel tie, or sleeper, that would meet all railroad-requirements and, at the same time, be able to compete in cost with the wooden ties now in general use. To meet these conditions it was necessary to use worn scrap-rails for the raw material, since it must be obtainable at a low price and in large quantities.

First, I designed sections for a steel tie, which railroad engineers said were ideal, provided it could be rolled. Then I designed a mill, shown in Figs. 7 and 8, to change the dimensions of a rail without lengthening it in the rolling. This mill can be driven like an ordinary two-high plate-mill, or by disconnecting the top roll from the power, it may be run

by frictional contact with the bar. The latter plan I consider to be the simplest and the best, for the reason that access

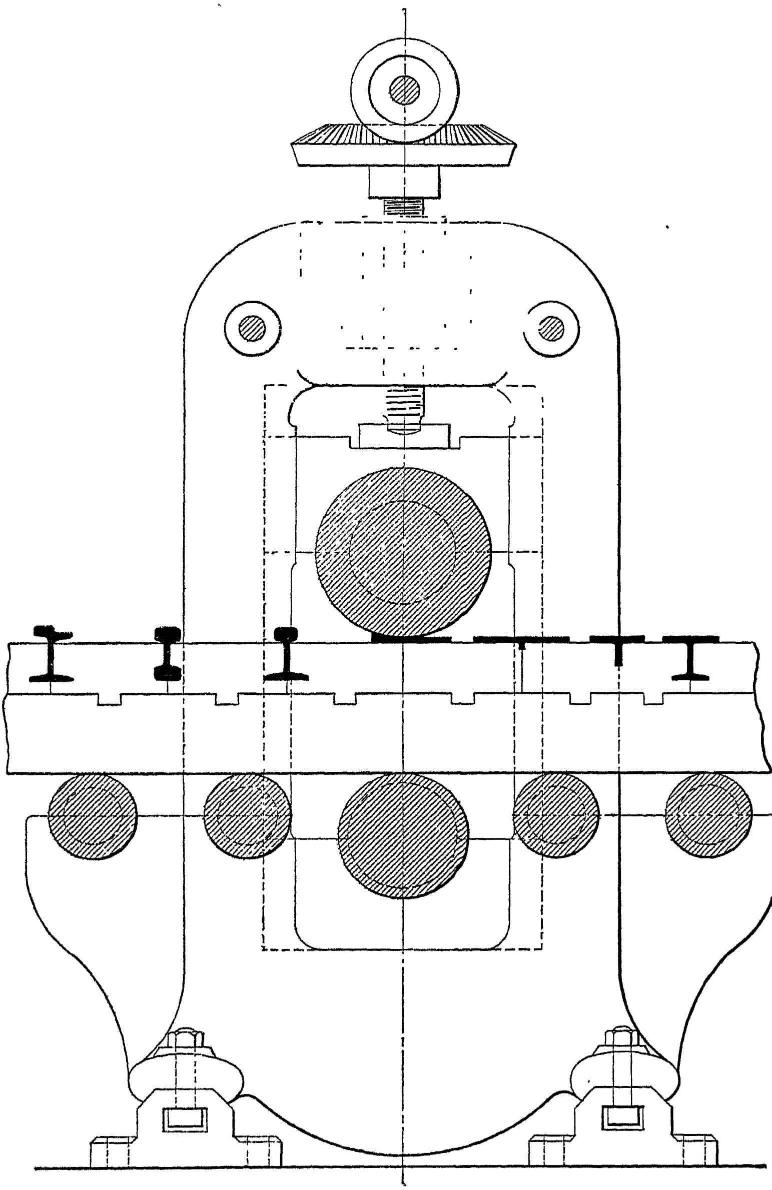

FIG. 7.—YORK TRANSVERSE MILL.

to the mill can be had in all directions, by having a hydraulic motor connected with a rack in a pinion on the bottom roll. Both plans have given good results. The simplest method

of all is to connect a hydraulic motor between two mills, rolling in both directions, which naturally increases the output.

By all other methods the greatest difficulty in longitudinal rolling is to produce sections having wide flanges; but in my mill this kind of rolling is very simple. One part of the bar is rolled without disturbing the other dimensions. Rails that

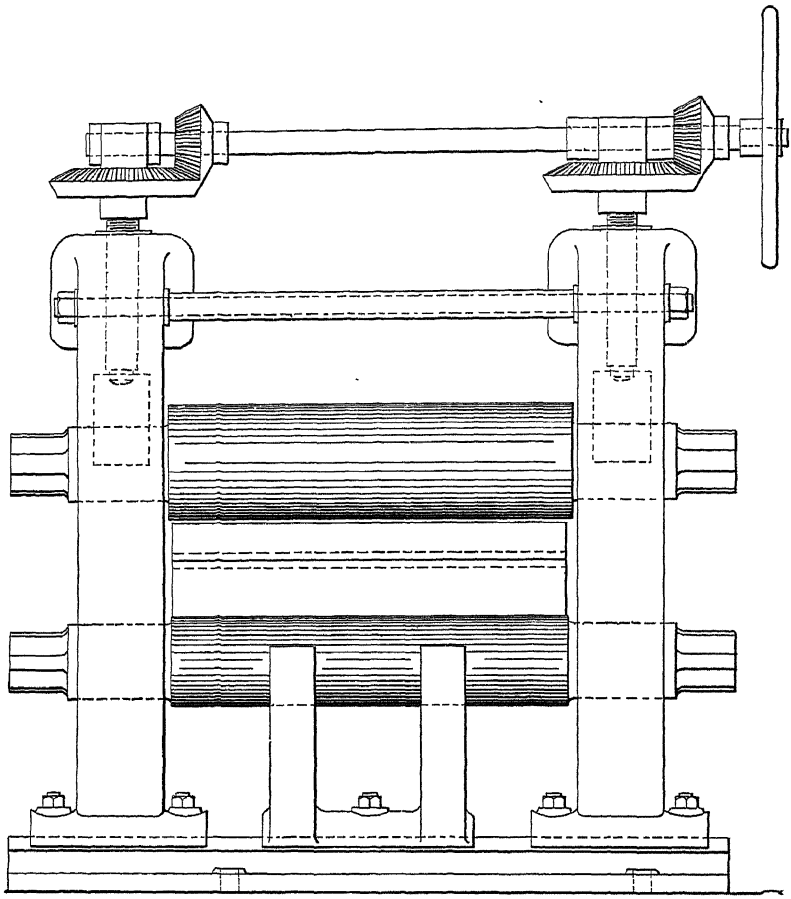

FIG. 8.—YORK TRANSVERSE MILL.

have been used on curves and have lost one-third of the metal on one side of the head, can be rolled to form a section having the metal symmetrically distributed on both sides of the web, by using a rolling and bending action simultaneously in the mill, as is illustrated in Fig. 9.

In rolling steel rails by the ordinary process, the web and bottom flange receive the most physical work in the mill, and

are finished much colder than the comparatively large mass of metal in the head. The head, therefore, is the weakest part of the rail. In re-rolling an old rail by my process, the head receives the greatest amount of physical work, and the finishing-passes can be made at a low heat, if desired, thus greatly improving the quality of the metal.

The capacity of the mill, in tonnage, would be regulated by the number of bars rolled at one operation. There is no difficulty in rolling at least six bars at a time, which gives a safe estimate of from 150 to 200 tons per day, of rails weighing from 60 to 75 lb. per yard.

During the rolling of any section in this mill, the flanges can be made thick or thin without any change of rolls.

This mill is confined to the transverse rolling of lengths not exceeding, say, 10 ft. If greater length is desired, a bloom (either of scrap rail or new steel) is rolled to proportionate di-

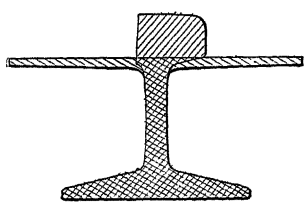

FIG. 9.—SHOWING THE RESULTS OF TRANSVERSE ROLLING.

mensions of cross-section, and then rolled to length in our universal mill.

Fig. 10 illustrates the mill rolling tees, tie-plates, column-blanks and double-headed rails. Fig. 11 shows the head of a worn 95-lb. girder-rail, having a part of the web attached, which is rolled to a width of 20 by $\frac{1}{8}$ in. Fig. 12 shows a T-rail section with the head rolled five times wider than its original dimensions. Fig. 13 shows a worn 70-lb. T-rail rolled down to a plain tie-section without changing the web or flange of the original rail. Fig. 14 shows the same weight rail rolled with a curved base to give elasticity to the tie under moving load. Fig. 15 shows the section with a curved seat for the rail, so as to insure the elasticity of the tie if the ballast in contact with the curved base should interfere with the elasticity of the lower flange. Fig. 16 shows a beam with flanges 9 in. wide and 3.25 in. in height of section, rolled from a 70-lb. rail.

Fig. 17 shows a section rolled with wide flanges, leaving the web without change of form. Fig. 18 shows a flange rolled on

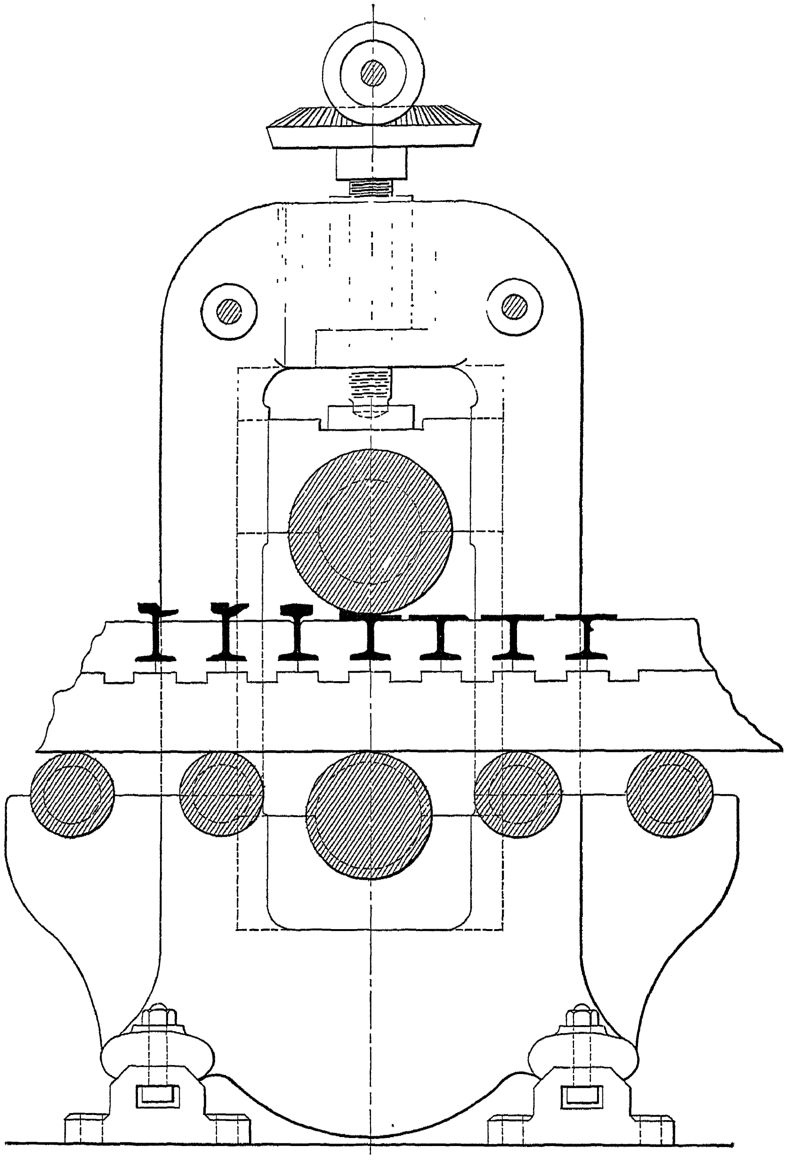

Fig. 10.—York Transverse Mill.

a 3- by 0.75-in. flat bar to six times the width of the thickness of the flat. Fig. 19 shows a column-section rolled from a 70-lb. rail. Fig. 20 is an end elevation of this mill, which

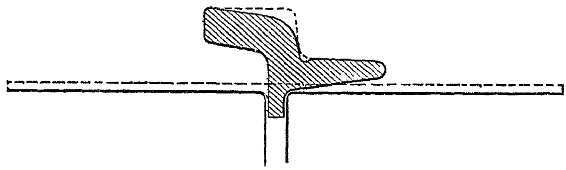

Fig.11

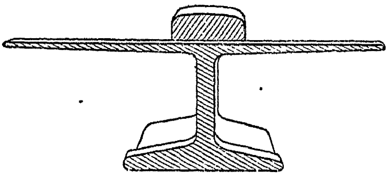

Fig. 12

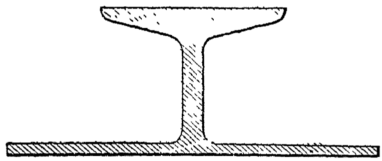

Fig.13

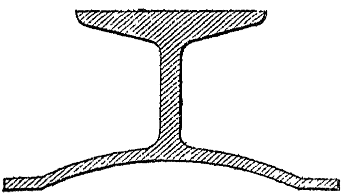

Fig.14

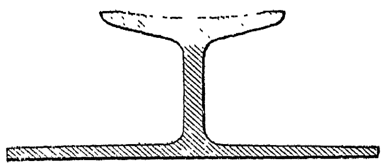

Fig.15

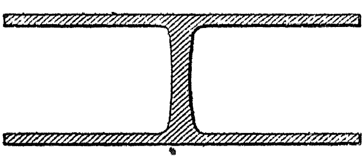

Fig.16

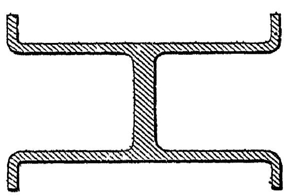

Fig.17

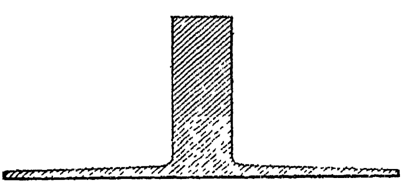

Fig.18

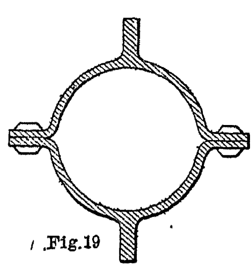

Fig.19

FIGS. 11 TO 19.—SPECIAL SECTIONS ROLLED BY TRANSVERSE MILL.

shows the rolling of flanges on large and small ingots and flat slabs for blanks for the finishing-mill. This arrangement dispenses with the unequal displacement of metal to form the web, and removes the greatest difficulty in the rolling of flange sections, and makes possible the production from new steel of sections that it is now impossible to produce by present methods.

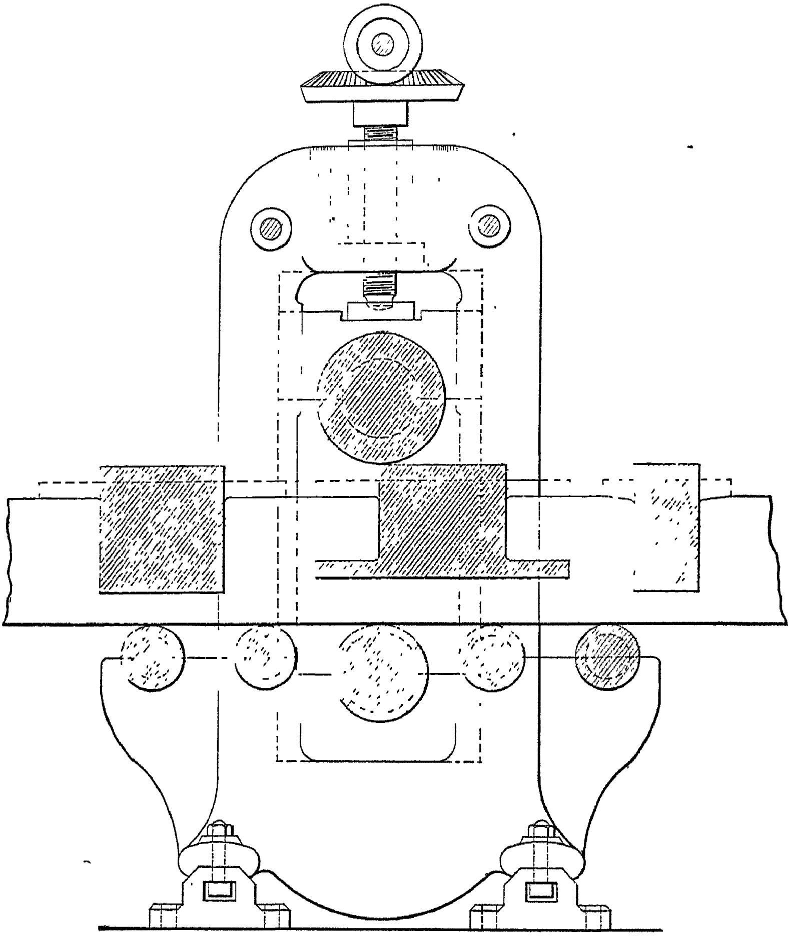

FIG. 20.—YORK TRANSVERSE MILL FOR ROLLING FLANGES ON INGOTS.

Fig. 21 shows some of the various sections, rolled from rails, to be used for columns, telegraph-poles, trolley-poles, beams, fence-posts and other purposes. To produce these sections it is necessary, first, to roll in a transverse direction to get the proper dimensions, and then to bend in the mill in a longitudinal direction.

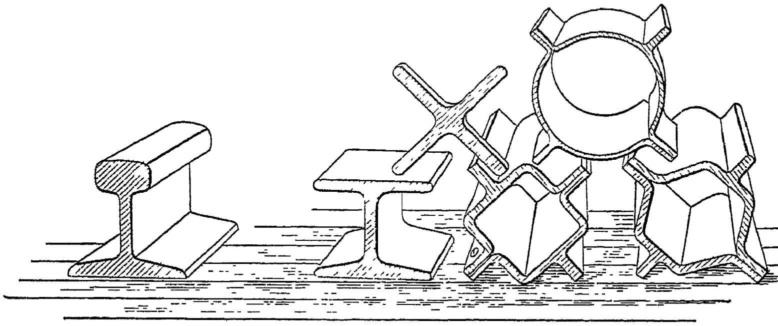

Fig 21

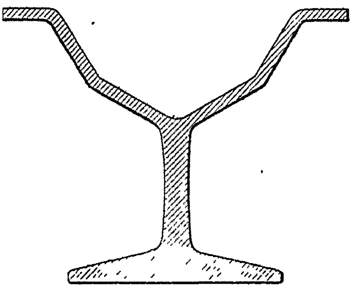

Fig.22

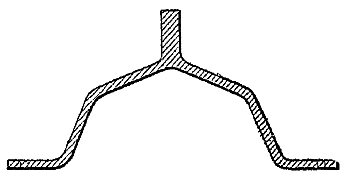

Fig.23

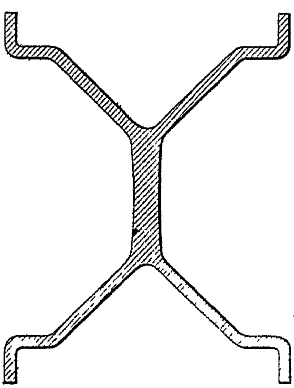

Fig.24

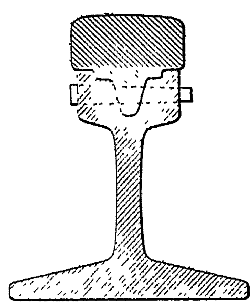

Fig.25

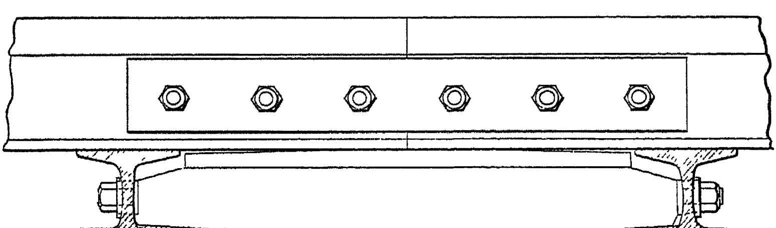

Fig.26

FIGS. 21 TO 26.—SECTIONS ROLLED BY TRANSVERSE MILL.

The following shapes can be rolled by using the mill for transverse rolling and longitudinally for bending hot: Fig. 22, a rail with the flange and web unchanged from the old rail, and the head rolled into the shape given. Fig. 23, a similar section rolled with part of the web removed. Fig. 24, a section where the rail-head and flange have both been rolled to the necessary width and then bent to the shape longitudinally. Fig. 25, a suggested base for a rail with a movable head. Fig. 26 shows a side view of a rail-joint, made from short pieces of worn rail. This joint is designed for the purpose of meeting

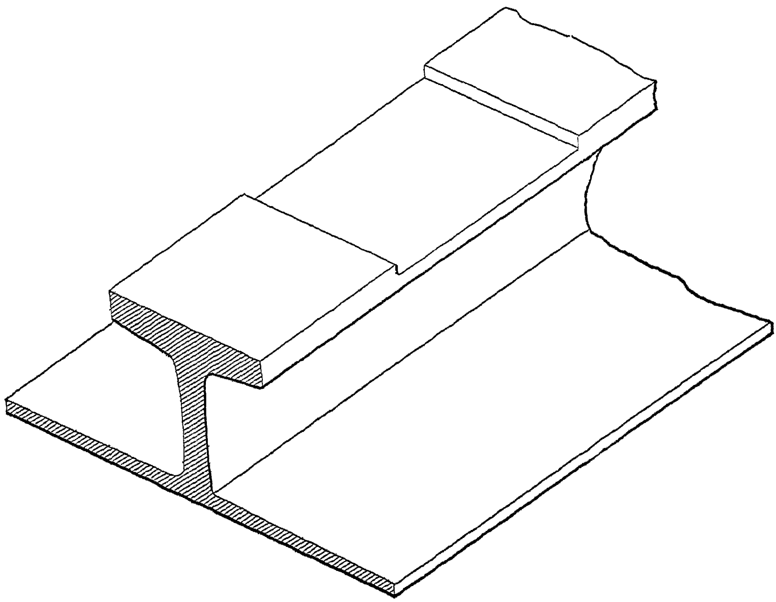

FIG. 27.—TIE-SECTION WITH RECESS FOR RAIL, ROLLED BY TRANSVERSE MILL.

any demands of adjustment, strength and cheapness. Fig. 27 shows a tie with a recess rolled to form a rail seat so as to prevent track-spreading. Fig. 28, column-sections with rolled projections parallel to the length. These can be changed, if desired, into indentations or any other shape. Fig. 29 shows taper sections of narrow or great width. These could be used for many purposes. Fig. 30 illustrates the utilization of short pieces of rail to make the tie-plates now generally used with wooden ties. I show one only, but I have already designed eight sections that can be made by this method.

All the sections described above, with the exception of the ingots shown in Fig. 20, are designed to be made from old worn rail-sections. The girder- and guard-rails of electric and street-car lines can be used to make the same class of material; and while the section would be of greater height, it is capable

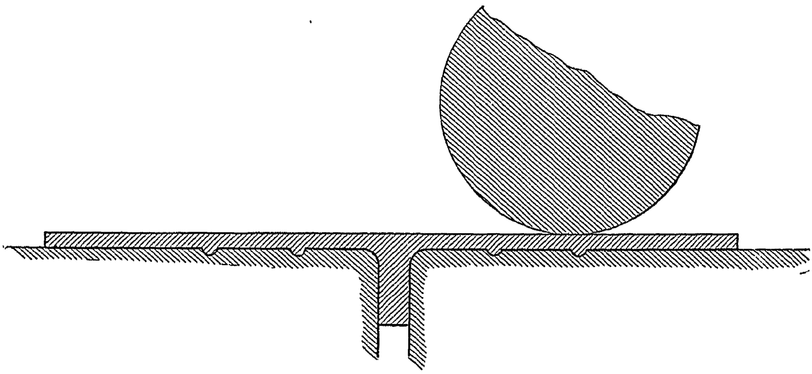

FIG. 28.—COLUMN-SECTION WITH PROJECTIONS PARALLEL TO LENGTH, ROLLED BY TRANSVERSE MILL.

of being rolled with the same facility as worn tee- or double-headed rails.

The method is not confined to the use of rail-sections alone as material for re-rolling. We take beams of all sizes, split the

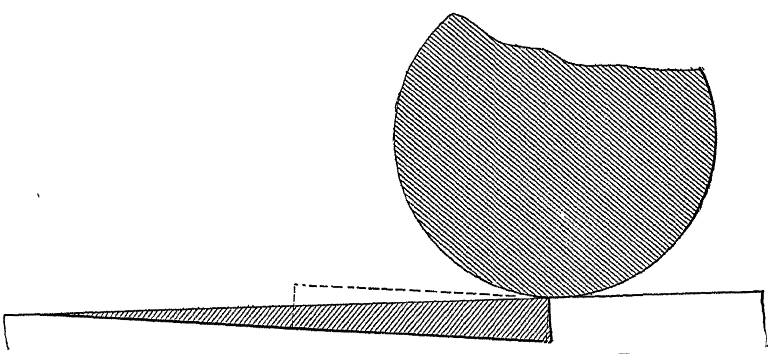

FIG. 29.—TAPER SECTIONS, WIDE OR NARROW, ROLLED BY TRANSVERSE MILL.

two flanges off, hot, in the machine, and use them for T-sections, column-blanks and other purposes after re-rolling. Angles, and, in fact, bars of any kind, are similarly available. The presence of holes in scrap material does not interfere with rolling by this method.

Some of the many sections which can be rolled by this process are shown in the accompanying illustrations. I have already designed several hundred shapes that differ from existing sections, either in weight or design.

It is adapted for flanging plates and sheets, hot or cold, and for making angles from flats; also, for re-rolling plates and sheets with decorative designs on their faces, for artistic and industrial purposes.

In planning these mills I kept in mind the necessity of making them strong to resist the hard usage of rolling-mill prac-

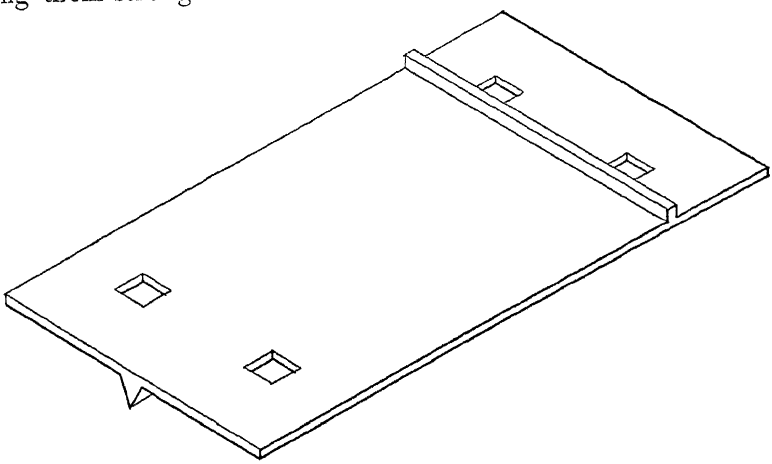

FIG. 30.—TIE-PLATE, FROM SHORT PIECE OF RAIL, ROLLED BY TRANSVERSE MILL.

tice; simple, so that they could be readily adjusted; and, also, capable of operating with unskilled labor, except that of the mechanic to run the machine and the heater to heat the material. The rails can be fed into the machine simply and quickly, and, when the rolling is completed, the hot bars push out, laterally, those already rolled. The functions of the machine are almost entirely automatic.

The economic and commercial importance of the application of this method to the utilization of scrap for ties, joints and tie-plates has been well stated, as follows:[2]

"At the present time there are about 450,000 miles of railways constructed in the principal countries of the world. Of this mileage probably 33 per cent. at least is double, counting sidings, which would raise the total, as single mileage, to 600,000 miles. The weight of metal required in any system of iron or

[2] *Iron and Coal Trades Review*, vol. lii., p. 514 (1896).

steel sleepers will, of course, depend mainly on its form and method of application; but a conservative mean would probably be 100 tons to the mile of single line, which would give us, for the railways of the world, a possible supply of 60,000,000 tons, or, roughly, about 40 times the total output of Bessemer steel in Great Britain during each of the last three years. Clearly, therefore, if metallic sleepers were substituted for timber, the steel trade would profit enormously."

Since the above was published, the *Archiv für Eisenbahnwesen*[3] reports an increase in mileage, at the end of the year 1903, to 534,292 miles, which, including side-tracks, is equivalent to about 700,000 miles. Estimating on a basis of 125 tons of steel per mile of single track, the total quantity of steel used reaches the enormous total of 87,500,000 tons.

If, as I have attempted to show, these rolled bars can be made from material that has already fulfilled the purpose for which it was originally made—namely, its use as rails—a great economic advance will have been effected.

The Tin-Deposits of the Kinta Valley, Federated Malay States.

BY WILLIAM R. RUMBOLD, ORURO, BOLIVIA, SO. AMERICA.

(London Meeting, July, 1906.)

THE Kinta valley, in the State of Perak, one of the largest of the Federated Malay States, is probably at the present time the richest alluvial tin-district in the world, Perak producing from 20,000 to 25,000 tons of tin annually, and the Kinta valley being the chief producer.

The valley runs approximately north and south, and is 30 miles long by 12 wide. It is very flat, the mountain-ranges on either side rising abruptly from the plain, that on the east being the great Central range of the peninsula, and that on the west a subsidiary range marked "Kledang" on the map, Fig. 1.

The valley is drained by the Kinta river, which rises in the eastern mountain range and flows south to join the Perak river, being joined by numerous streams from both mountain-ranges.

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[3] *Railroad Gazette*, vol. xxxviii., p. 749 (1905).

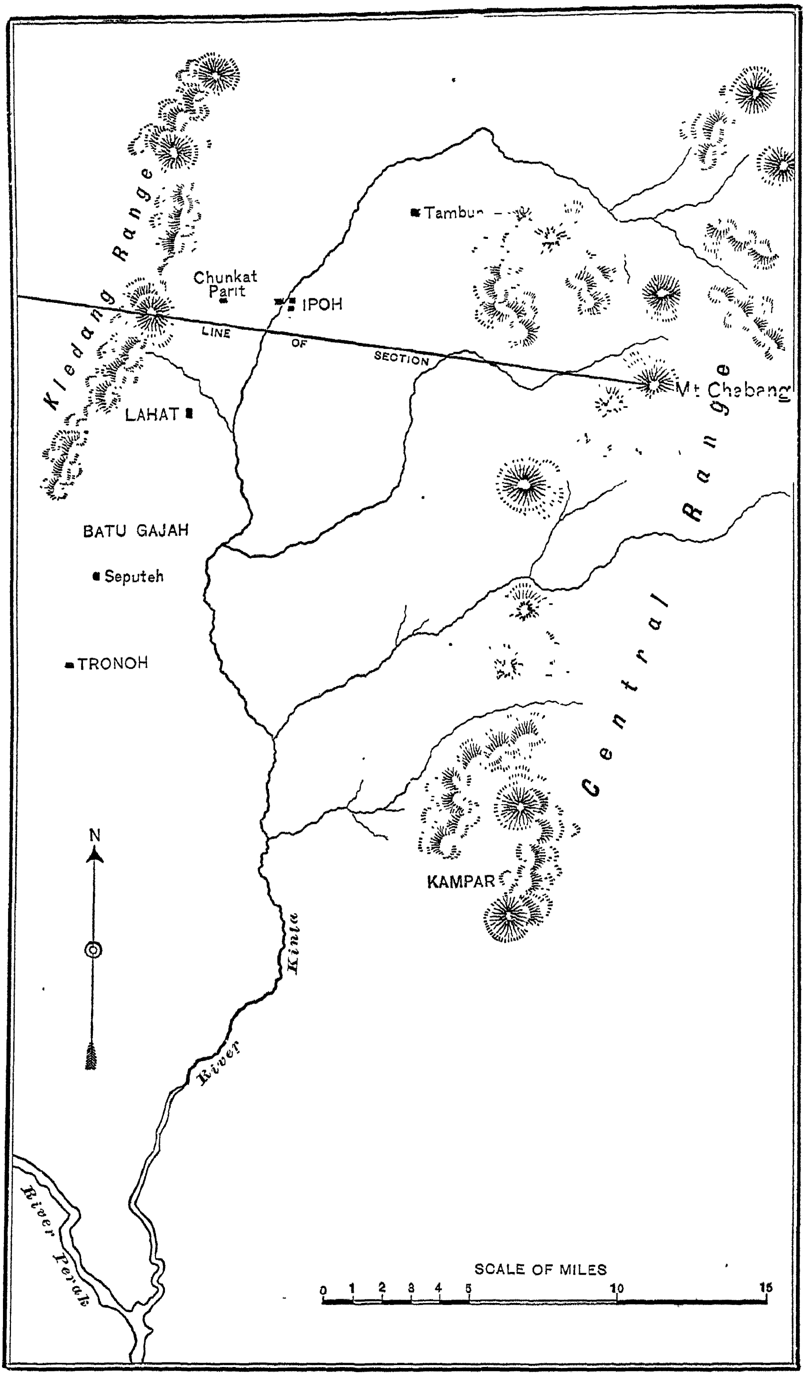

FIG. 1.—SKETCH-MAP OF THE KINTA VALLEY, FEDERATED MALAY STATES.

The rain-fall is heavy, averaging annually about 90 in. The whole country is covered with dense jungle, except where it has been cleared for mining-operations and a small amount of cultivation.

Mining villages are scattered throughout the valley, the chief towns being Kampar, Batu Gajah, Lahat and Ipoh, the latter being the center of the mining-operations.

An ideal section of the main geological features of the Kinta valley is given in Fig. 2. The valley is composed of a highly crystalline limestone, usually white in color, sometimes gray; in fact, it may be called marble. It is always highly inclined and often contorted, and in some places is interbedded with shale. On the east side of the valley, near its contact with

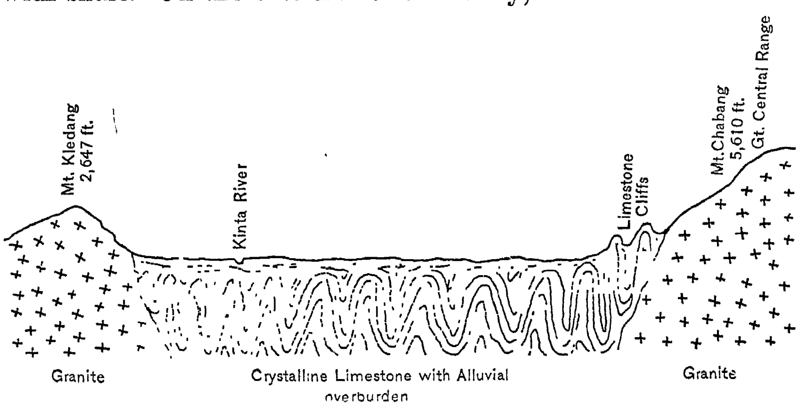

FIG. 2.—IDEAL SECTION OF THE KINTA VALLEY.

the granite, it forms a remarkable series of limestone cliffs, which rise in some cases as high as 2,000 ft. above the level of the limestone in the valley. It is non-fossiliferous, and its geological age is unknown. The mountains on both sides are composed of granite, which is intrusive and has doubtless been the cause of the contortion and metamorphism of the limestone, the latter appearing as though it had been squeezed between the two intrusive masses. The granite is gray in color, and often porphyritic, with large, well-formed crystals of orthoclase-feldspar, quartz, and biotite-mica. Tourmaline is very common.

The whole valley and a large proportion of the mountain slopes are covered with alluvium, which reaches a depth of 200 ft., as proved by mine-workings, and may be deeper in the middle of the valley, where there are very few workings; the

average depth is about 30 ft. This alluvium is composed of sand, pebble-beds, and clay, and may rest either on a bed-rock of limestone or of "kong," the latter being generally accepted as china-clay, but, as Mr. J. B. Scrivenor has shown in his paper,[1] may be also the bleached surface of shales.

The limestone, from its steep bedding-planes and contorted structure, has been weathered to a very irregular surface, forming numerous little pinnacles and depressions, the latter often extending into cracks of 10 to 20 ft. in depth, the whole forming an ideal series of natural riffles.

Alluvial Tin-Deposits.

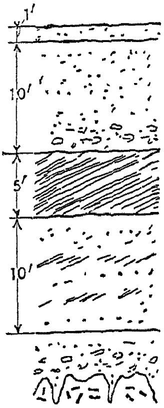

Fig. 3.—Section of Open-Cast Mine near Seliben.

The alluvial tin-deposits are exceedingly variable, following no well-defined "lead"; the "karang" (tin-bearing wash) may be sandy, or the tin may be carried in a thick, heavy clay; there may be one or two layers of karang, or the whole soil may carry tin, from bed-rock to grass-roots. The section presented in Fig. 3 of an open-cast mine near Seliben may serve as perhaps a typical deposit. The tin occurs as small black crystals, usually, but not always, water-worn, mixed with tourmaline, ilmenite, and other impurities, which will be referred to later. Another interesting and frequent variety is the occurrence of tin in heavy red ferruginous clays, which often carry values throughout their depth. One of the most striking instances of this class is the Tambun mine, 4 miles NW. of Ipoh, which, at the time of my visit to the district in 1903, was the richest mine in the valley, producing 350 tons of concentrates a month, assaying 68 per cent. of metallic tin. The average value has been calculated from the number of cubic yards washed in a month and the amount of concentrates obtained; but a sample taken by me from a number of car-loads gave a higher result—76.8 lb. per cu. yd. Fig. 4 shows a section of the deposit.

The deposit was situated at the top of a small rise and appeared to be completely isolated, boring operations which had

[1] *A Preliminary Report on the Geology of the Neighborhood of Taiping.* (Published as a Supplement to the *Perak Government Gazette,* Jan. 15, 1904.)

been energetically carried on in the vicinity failing to prove an extension of the deposit.

The red and black karang carried a considerable quantity of iron oxide and some manganese: the tin often occurred in the boulders and lumps of ironstone mentioned above; indeed, the mine had erected a special plant to deal with them. The cassiterite was much less water-worn than in many of the alluvial deposits (good crystals being by no means uncommon), and ranged in size from lumps as large as a man's fist down to the finest sand. The granite contact is 2 miles from the mine.

Another well known and almost equally rich mine is at Tronoh, 9 miles SW. of Batu Gajah. This mine is situated at the bottom of a small hill, and is supposed to be about 200 ft. deep; when I visited the mine, June, 1903, bed-rock had not been reached. The overburden of about 50 ft. consists of red and yellow sand and clay, the karang being a heavy blue clay, in which appear white water-worn pebbles of quartz; the latter are supposed to be characteristic of the places where the richest tin occurs.

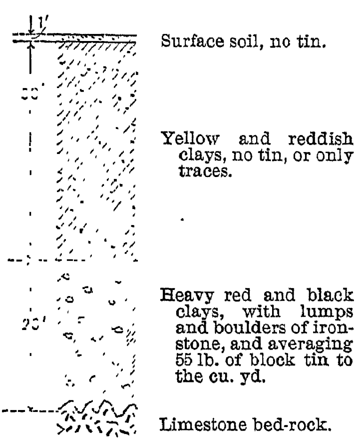

FIG. 4.—SECTION OF TAMBUN MINE.

The bed-rock, although not exposed, is almost certainly limestone, which occurs in the vicinity and also in shafts that have been sunk close to the deposit. The cassiterite is considerably more water-worn than at Tambun, and is found as clean black crystals, without any admixture of iron oxide; the concentrates, however, contain a small amount of iron pyrites.

There are numerous other mines that might be described, notably those on the slopes of the mountains, often on granite bed-rock, generally low in tin values and much mixed with tourmaline, ilmenite and granite decomposition-products. There is also a very interesting series of deposits in the hollows of the limestone cliffs mentioned above; these deposits, often 500 ft. or more above the level of the valley, are invariably found

in a red ferruginous clay, containing water-worn pebbles of iron oxide. The values are not often high, but the tin runs throughout the deposit, being, however, richer near bed-rock. These last-named deposits afford some measure of the enormous time that the present denudation of limestone and granite and the concentration of their heavier particles has been taking place, and awakes speculation as to the amount of concentration and reconcentration that has been going on since the cassiterite was first weathered out fo the rock.

The minerals met with in the wash include magnetite, tourmaline, ilmenite, iron pyrites, wolframite, garnet, zircon, spinel, corundum, and in one mine an interesting concentration of cerussite.

The tourmaline, ilmenite, zircon and garnet occur more abundantly near the granite contact, the tin in these mines being free and in black crystals; in the red clay-deposits these minerals are rare, and generally entirely absent, the tin in the latter cases being found with iron oxide.

THE LODE DEPOSITS (in contradistinction to alluvial deposits):

(1) *Those in Granite.*—Most writers on the alluvial deposits of the Malay Peninsula state that the tin of the alluvium must have been derived from stockworks or lodes in the granite, yet can point to but very few instances. This failure to find the "root-deposits" may be ascribed to the thickness of the alluvial covering and to the denseness of the jungle; but surely such sweeping assertions as are sometimes made should be based on stronger evidence than hitherto has been adduced. The only lode in the granite, or indeed any tin-deposit in that rock, that I have seen in this district was one west of Lahat, at the bottom of the granite range and near the contact with the limestone. This lode was about 2 ft. wide with no well-defined walls, the lode-stuff consisting of an intimate mixture of gray cassiterite and iron pyrites with a little quartz; it appeared as though the minerals had replaced the original feldspar in the granite, and the deposition had taken place along a joint-plane in the rock.

(2) *Those in Limestone.*—There seems to be a general belief that the occurrence of cassiterite in limestone is, if not an impossibility, so rare that the discovery of profitable lodes in that

rock is hardly worth discussion; but if limestone is in connection with intrusive stanniferous rocks there appears to be no sound reason against the occurrence of tin in it, just as plentifully as in slate rocks placed in a similar position.

In the Kinta valley there are at least three deposits of tin in limestone; probably more, but the difficulty of obtaining any reliable information as to deposits that have been abandoned and are supposed to be worked out, makes it often impossible to be sure whether a mine was alluvial or otherwise.

The three deposits actually seen by me were those of Chunkat Parit, Saiah, and a mine owned by the French Mining Co. near Lahat.

In Chunkat Parit, one mile west of Ipoh, at the time of my visit, the lower workings were under water, and I had to be content with a look at the outcrop, the dump, and specimens. The outcrop could not be traced for any great distance, and the lode did not show signs of well-defined walls, but appeared to follow roughly a bedding-plane in the limestone, striking N–S., and apparently rapidly thinning out at each end. The lode-stuff was a mixture of brown fine-grained cassiterite, erubescite, and chalcopyrite, with a limestone gangue. The country-rock was the usual highly crystalline limestone, the deposit being about two miles distant from the granite contact.

Saiah, close to Seputeh, SW. of Batu Gajah, is an open-cast mine worked by a Chinaman. Unfortunately, here, too, I was unable to make a close examination, not being allowed to enter the workings; but it appeared that the mine had been originally worked for fairly rich alluvium, and when this had been followed down into one of the cracks of the limestone, a lode deposit was discovered. The lode-stuff consisted of cassiterite, arsenical pyrites, iron pyrites and quartz, some of which had been deposited in well-developed crystals. The country-rock was limestone, the deposit, as far as could be seen, following the general strike of that rock. It was two miles from the nearest granite outcrop known to me.

The third deposit, situated a little east of Lahat, and owned by the French Mining Co., was by far the most interesting and most instructive.

The discovery had only recently been made, when I was kindly given the opportunity of examining it. The lode had

been stripped of its alluvial covering for 50 ft. along the strike; whether the alluvium had been rich in tin, and whether this fact had led to the discovery, I am unable to say. The lode was 8 ft. wide with a NE–SW. strike, and dipped at a steep angle to the west. It was composed of an intimate mixture of red iron oxide and cassiterite with a little iron and copper pyrites, the gangue being calcite and limestone, while there were some very large and beautiful crystals of calcite on the hanging-wall. The walls were fairly well defined, much more so than in either of the deposits previously described. The granite contact was a mile to the west of the mine.

I much regret that I have no details as to the subsequent developments; the company were starting to prove the existence or otherwise of the lode in depth by a core-drill.

Origin of the Deposits.

It appears to be generally accepted that the alluvial tin has been derived from original deposits in the granite, and as proof thereof there are cited the few lodes that have been discovered, the analogy to the Erzgebirge and other tin-fields, and sometimes also the fact of the constituents of the granite being found with the tin in the karang,—this latter a most remarkable statement. One might just as well argue that, because one of the mines mentioned above contained cerussite, obviously eroded from the limestone, therefore the minerals mixed with the cerussite, such as quartz, tourmaline, feldspar, cassiterite and even granite pebbles themselves, had been derived from the limestone.

The absence of certain minerals, however, may sometimes be of use in determining the origin of the alluvium.

At the same time, although I think that in a few cases it can be proved otherwise, it is possible and even probable that a great many, perhaps a great majority, of the tin-deposits have been derived from the granite. There is the proof of the tin-alluvium found on the slopes of the hills and on granite bed-rock, in some cases far away from and above the limestone, which must have been derived from the granite itself; it is also noticeable that the sides of the valley are richer than the center, in which place mines are conspicuous by their absence.

But in the case of such a deposit as the Tambun mine, 2

miles from the granite, the karang rich in iron oxides and manganese, the cassiterite only slightly water-worn, the occurrence of the latter in ironstone in large blocks, the isolated character of the deposit, and the failure to trace any "lead," would naturally suggest its local origin; and when to this is added the actual occurrence of tin in limestone, especially in the case of such a deposit as that described at Lahat, which, with its intimate association of iron oxide and tin, would tend to produce just such an alluvium as is seen in the Tambun mine, it is for me an impossibility to go back to granite for its origin. It is perhaps superfluous to add that, because a deposit such as the one referred to has been derived from a lode in the limestone, it does not necessarily follow that the root-deposit can be discovered, since the alluvium was very probably the result of the disintegration of rich pockets, the ores of which perhaps arrived through fissures, which at the present plane of erosion have no marked distinguishing features.

Mr. Scrivenor, in his paper already referred to,[2] writes:

> "So far, I have seen nothing in the ironstone which cannot be accounted for by the deposition and subsequent oxidation of iron carbonate conveyed in solution by percolating water. That the water should find its way along the bedding-planes is not surprising, seeing the high angle at which these planes are placed."

Again he says:

> "After the deposition of the alluvium percolating water carrying iron carbonate in solution caused the formation of the cemented karang and the laterite."

I do not know whether Mr. Scrivenor would call the Tambun karang laterite or not, but it seems to me a little difficult to imagine ascending solutions of iron carbonate circulating through the upper part of the limestone without precipitation, and then forcing their way through a most impervious stratum of alluvial clay. Surely, in alluvium, if anywhere, we must find rather descending oxidizing solutions, and I should be much more inclined to attribute the laterite and cement-beds to the solution of iron oxides from the granite and limestone by organic acids derived from decaying vegetation, and subsequent precipitation of the iron thus carried. Nowhere in the world would such organic acids have a better opportunity

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[2] *A Preliminary Report on the Geology of the Neighborhood of Taiping.* (Published as a Supplement to the *Perak Government Gazette*, Jan. 15, 1904.)

of forming than in the moist climate and dense vegetation of the Malay States. The bleaching of the shales to which Mr. Scrivenor alludes may perhaps be accounted for by the acids formed in this way.

The Tronoh mine appears to be as isolated as that of Tambun, but there do not seem to be such definite proofs of its derivation from limestone as in the latter; on the other hand, definite proofs of its derivation from granite appear to be equally lacking.

There are many other mines, especially those which are some distance from the granite contact, the origins of which seem referable rather to deposits in the limestone than to those in the granite; that is, given the possibility of the occurrence of tin in the limestone, which is a fact beyond dispute. With regard to the concentration of the cassiterite, especially in explanation of the occurrence of two or more layers of karang, I think that Posepny's theory[3] is excellently illustrated—namely, that of the sinking of the heavier constituents through a loose mass with the aid of water until the particles reach bed-rock or a relatively impervious stratum.

In every case that I noted, where there were two distinct beds of karang, the top bed was invariably lying on clay; sometimes the clay itself was a tin-bearer, as though the heavy particles had endeavored to work their way through the close-packed bed, but had found the process so difficult that they had become concentrated there.

As regards the origin of the "root-deposits" themselves, there seems for the granite no reason to depart from the usually accepted theory of tin-deposits—namely, that they are formed by the after-actions of the eruptive rock; but in the limestone it would seem more in accordance with the facts to suppose that the lodes were formed by ascending waters, doubtless closely connected with the pneumatolytic after-actions.

In conclusion, I desire to remark on the extraordinary indifference which is, or at any rate until lately had been, displayed by the Government of the Federated Malay States in matters geological, and on the extreme reticence which characterizes the Department of Mines, even as to the small amount of information which they have collected.

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[3] *Genesis of Ore-Deposits*, 2d ed., p. 154 (1902).

It is not too much to say that at present the country is absolutely dependent, for its progress and for its commercial and financial existence, on the tin industry. The tax on tin pays for the roads, the railways, the many fine government buildings, and the salaries of those that work in them; and yet it is only quite recently that the Government has appointed a State Geologist. In a mining country in which mines are so quickly discovered, worked out and abandoned as in the Kinta valley, it seems of enormous importance that careful records of each mine should be kept, and such points as the character of the overburden, karang, and bed-rock, the accompanying minerals, the approximate richness of the karang, etc., etc., should be noted, correlated with others, and published, together with borings or any other exploratory work which may be undertaken by the Government. That one man, however able, can collect these data as well as fulfill his more strictly geological work, is of course impossible; but the mining inspectors, many of whom are technically trained men, might certainly make use of their exceptional opportunities in this direction, and aim at being something more than "Guardians of the Water," as the Malays and Chinese now call them. In such ways we might be able to learn more of these unique and most interesting deposits, and not only speculate as to their origin, but be able to point to paying lodes as evidence of our theories.

To quote Posepny's famous work again (p. 3):

"Mining, indeed, constantly furnishes fresh evidences in new openings, but it destroys the old at the same time; and if these are not preserved for science before it is too late, they are lost forever. The whole mining industry is in its nature transitory; but the nation, which intrusts to the miner, upon certain conditions, the extraction of its mineral wealth, has a right to demand that the knowledge thus gained at the cost of a part of the national resources shall not be lost to science."

Fluorite and Barite in Tennessee.

Author's Postscript to a Paper on "Lead- and Zinc-Deposits of the Virginia-Tennessee Region," *Trans.*, xxxvi., 681 to 737 (1906).

BY THOMAS L. WATSON, BLACKSBURG, VA.

MY thanks are due to Mr. Frank Firmstone, Easton, Pa., who has called my attention to the statement in my paper[1] that "Barite, fluorite and quartz, though not observed in the Tennessee area," . . . as possibly misleading to the reader, if construed to mean that such a discovery had never been made. Mr. Firmstone is correct in assuming that this statement referred only to the study of the worked areas of the metallic ores, especially zinc, made in the course of the survey upon which my paper was based.

The occurrence of barite and fluorite with lead-ores in Tennessee was reported by Prof. Safford,[2] in 1869, and by Safford and Killebrew,[3] in 1887, with the further statement that about 1,000,000 lb. of barite were annually mined in Greene, Washington and Jefferson counties, Tenn., and that considerable deposits occur also in McMinn, Smith and other counties. It is also noted that the barite "usually occurs in veins, associated with galena." The present worked areas of barite are, however, distinct barite propositions, without workable quantities of either lead or zinc, so far as mining operations have gone, and the areas are more or less removed from the present areas of zinc-ores.

Mr. Firmstone informs me also that, in 1892, he observed, in the magnesian limestone at Watauga Point, in Carter county, on the left bank of the Watauga river, a little below the mouth of Buffalo creek, small quantities of galena and some fluorite.

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[1] *Trans.*, xxxvi., 686 (1906).

[2] *Geology of Tennessee*, by James M. Safford, A.M.,Ph.D., p. 224, Nashville (1869).

[3] *Elementary Geology of Tennessee*, by James M. Safford and J. B. Killebrew, pp. 140, 198, Nashville (1887).

DISCUSSIONS.

The Secondary Enrichment of Copper-Iron Sulphides.*

A Discussion of the Paper of T. T. Read, p. 297.

E. C. SULLIVAN, Washington, D. C. (communication to the Secretary†):—Mr. Read's paper is deserving of high commendation for attacking the problem from the experimental side, yet certain errors have crept into the work to which it seems desirable to direct attention. Most serious of these is the fact that in the two experiments in which data are given, showing the total quantities and composition of solid and solution before and after the experiments, there are discrepancies great enough to deprive the results of significance.

In Experiment No. 1 the author has, before the experiment:

10 g. of chalcopyrite containing	3.262 g. of copper and 2.724 g. of iron.
100 cc. of solution containing	0.360 g. of copper. ......
Total,	3.622 g. of copper and 2.724 g. of iron.

At the conclusion of the experiment:

9.54 g. of chalcopyrite containing	3.234 g. of copper and 2.563 g. of iron.
100 cc. of solution containing	0.338 g. of copper and 0.343 g. of iron.
Total,	3.572 g. of copper and 2.906 g. of iron.

There is an apparent gain of 0.182 g. of iron, while solid and solution are each made to lose copper, the error, 0.050 g., being more than twice as great as the amount of copper (0.022 g.) which the author concludes was precipitated from solution by the chalcopyrite.

In Experiment No. 3, in which data are given for making the calculation, there are similar discrepancies, the iron appearing to increase from 2.724 to 2.827 g., and the copper to decrease from 3.262 to 3.241 g.

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* Published by permission of the Director of the United States Geological Survey.

† Received June 12, 1906.

The data as given by the author indicate on their face that the precipitation of copper from cupric sulphate solution by chalcopyrite is greater in the absence of sulphurous acid, a result not in keeping with the work of H. V. Winchell,[1] which seemed to show that the reaction between pyrite and cupric sulphate was facilitated by the presence of sulphurous acid. In experiments made by me to test this point, cupric sulphate solution (40 cc. containing 0.1 g. of copper) saturated with sulphur dioxide lost its color almost at once on shaking with 20 g. of finely powdered chalcopyrite, while cupric sulphate without sulphur dioxide, similarly treated, became colorless only at the expiration of a week. To each of the colorless solutions 4 cc. of a cupric sulphate solution was then added, containing 0.250 g. of copper. On standing over night, the solution containing sulphurous acid was again colorless, while the color in the other faded very slowly. In similar experiments with pyrite and cupric sulphate in presence and in absence of sulphurous acid, the copper at the end of three days had been practically removed from the solution which contained cupric sulphate and sulphurous acid, while from the solution which contained cupric sulphate alone about 0.040 g. of copper had been precipitated out of a total of 0.097 g. of copper in 40 cc.

There seems, therefore, to be no doubt that sulphurous acid accelerates the precipitation of copper, presumably as sulphide, from cupric sulphate solution by pyrite and chalcopyrite. The fact should be emphasized, however, that the presence of sulphurous acid is by no means essential to the precipitation of copper from cupric sulphate solution by these sulphides. Indeed, it may be open to question whether, so far as the quantity of copper precipitated is concerned, the difference caused by the presence of sulphurous acid is great enough to be of significance to the geologist.

Only one other point will be mentioned. The work of Morgan and Smith,[2] cited by Mr. Read as demonstrating that chalcopyrite is a cupric ferrous compound (CuS, FeS) rather than a cuprous ferric (Cu_2S, Fe_2S_3), seems not to be perfectly conclusive. Morgan and Smith, on passing hydrochloric acid gas

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[1] *Bulletin of the Geological Society of America*, vol. xiv., p. 269 (1902).

[2] *Journal of the American Chemical Society*, vol. xxiii., pp. 107 to 109 (1901).

over the mineral, and treating the residue with water, obtained a solution which reduced permanganate in the quantity to be expected if the iron were ferrous. Inasmuch, however, as cuprous chloride and ferric chloride may react to form cupric and ferrous chlorides, it would appear that the result might be the same whichever of the two formulas were correct.[3]

T. T. Read, Colorado Springs, Colo. (communication to the Secretary*) :—The discrepancies to which Mr. Sullivan adverts had not escaped observation; but the comparison he makes is somewhat unfair, as the errors lie in the determinations on the solid residue (it was not possible at the time to make them as carefully as was desirable), while the inferences as to the action which had taken place are properly drawn from the composition of the solution.

The chalcopyrite used for the original experiment was absolutely pure; no foreign material could be detected by most careful microscopic examination. Upon repeating the experiment upon ordinarily clean chalcopyrite, containing small amounts of other sulphides, results were obtained which agree with the experience of Messrs. Sullivan and Winchell. It would seem, then, that, under ordinary circumstances, the presence of sulphur dioxide accelerates the precipitation of copper from a solution of the sulphate.

In regard to the strictures upon the work of Morgan and Smith, it seems only logical to inquire, since the ferric and cuprous salts react upon the decomposition of the mineral to form ferrous and cupric salts, why they should not have similarly reacted at the instant of formation of the mineral. Allan and Gibb, and, more recently, Röntgen, have shown that intermetallic compounds of cuprous and ferrous sulphides exist in mattes. It would seem more reasonable to suppose that chalcopyrite is an intermetallic compound of cupric and ferrous sulphides rather than of cuprous and ferric sulphides, especially since the latter is not known as a product of natural reactions.

[3] Cf. Stokes, *Bulletin No.* 186, *U. S Geological Survey*, p. 45 (1901).

* Received September 28, 1906.

Improvements in Rolling Iron and Steel.

A discussion of the Paper of James E. York, p. 859.

ROBERT W. HUNT, Chicago, Ill.:—It has been my good fortune to know of this development of Mr. York's for some time, and I think he will permit me to say that this is not the first demonstration that he has made of his method for rolling difficult shapes, and which, as he has stated, had in the past been regarded as practically impossible. I have good reason to know that Mr. York has been successful on other lines, because I myself went through a period of professional disrepute for having reported favorably on his methods, which received discredit up to the time that it was practically demonstrated that his claims were true ones. Years ago Mr. York built a mill at Duluth, Minn., for the production of structural shapes, and notably of what might be called balanced beams—beams that did not have flanges of restricted width. His scheme was to produce beams with flanges so wide that they could be used in place of built-up beams. The mill, of course, went through the difficulties that such a mill would have to encounter. I was called upon to make an examination of the property and report. These beams were rolled on a universal mill—a mill which would admit of various-sized sections without change of rolls. I was convinced that Mr. York could do it, and I so reported. Having received condemnation and ridicule for having taken that position, later on I was glad to find that Mr. John Fritz had come to the same conclusion, and had so reported. I concluded we both could rest content, while awaiting developments. Later still, as we all know, there was another difficulty that Mr. York met with—namely, that the architects or structural engineers would not accept his proposed sections. That is still a great difficulty in America. Later, the Grey mill came into successful operation in Luxemburg, producing such sections, which are being used in England and on the Continent. In fact, the whole process, both from a commercial and a mechanical point of view, is a success. The Bethlehem Steel Co., Bethlehem, Pa.,

are putting in a Grey—that is, a York mill, as part of their immense new plant which is in course of construction. Owing to matters over which Mr. York had no control—that is, the financial side of the question—it was taken out of his hands, but it is the York mill that is making the sections. When Mr. York showed me the present examples of his skill, I was quite as much surprised as the company, but I was perfectly prepared to credit his claims.

KURT KERLEN, Düsseldorf, Germany:—I am of the opinion that the paper read by Mr. York would have been better if divided into two parts—*i. e.*, (1) Universal Girder Rolling-Mills; (2) Transverse Mills.

As I have only to-day for the first time heard anything of the latter type of mills, I am not able to say anything about them, but I think this idea of transverse mills, brought before us to-day by Mr. York, must be very interesting.

On the other hand, universal girder rolling-mills are not by any means a new idea, as on May 9, 1889, Mr. Hugo Sack, then of Duisburg, read before the Iron and Steel Institute a paper on universal girder rolling-mills for rolling girders and cruciform sections.[1] He gave in his paper an exact statement of the scientific and technical reasons why the ordinary method of rolling flanged sections in two-high and three-high mills is objectionable. At the same time he exhibited a model of his mill, which had been already working, among other works at those of Messrs. David Colville & Sons, Ltd., and the Newton works of The Steel Co. of Scotland, Ltd., and I take it from the discussion recorded in the proceedings of the year 1889 of the Iron and Steel Institute, that Mr. E. Windsor Richards was present at the discussion, and so that gentleman could corroborate me in what I have said. I may also say that at a later date this model was again exhibited in this country and that there exists another universal mill—the Grey mill—which is already in actual working at Differdingen.

The chief difference between the Grey mill, on the one hand, and the Sack and York mills, on the other, is that the Grey mill has a special pair of horizontal rolls, which are placed at some distance away from the four other rolls, and act

[1] *Journal of the Iron and Steel Institute*, vol. xxxiv., No. 1, pp. 132 to 145 (1889).

especially upon the edges of the flanges, while in the Sack mill this action upon the edges of the flanges is done simultaneously by the four rolls which are lying in the same plane. This is a great advantage and is much simpler when compared with the Grey mill.

As regards the York mill, the manner in which the compression of the flanges upon their edges is performed is not demonstrated in the paper, and from the short notes made by Mr. York, I do not at present see how he intends to roll girders with flanges of an exact width and with the web precisely in the middle of the flanges. I would like very much to hear further from him on this subject.

I am at the same time thankful to Mr. York for drawing the attention of this meeting to the universal system of rolling girders, which system without doubt will be of far greater importance in the future than it has been in the past. In fact, the method now in vogue of rolling girders—*i. e.*, in two-high or three-high mills—is a very crude one, as I will now endeavor to point out.

In order that a rolled bar may pass easily into the next groove, the latter has always to be made somewhat wider than the previous one, and consequently in this way sections are always obtained of increasing width, and it is obvious that under such circumstances no pressure can be exerted upon the outer surfaces of the flanges, and their widths cannot be much reduced. Further, only the inner surfaces lend themselves as a point of attack for reducing their thickness. By this process the rolls must evidently squeeze themselves between the inner surfaces of the flanges, and scrape the material down the inner face of the flange, the edges of the rolls forming there a ridge and accumulating material in the corners. This material must be transplaced laterally across the fiber, thus weakening the tensile strength considerably.

Accordingly one might divide the rolling of a girder in an ordinary mill into three distinct stages, viz.:

1. The rolls come into contact with the flanges and begin to scrape;
2. The material is piled and is displaced laterally;
3. The web takes its form.

Before acting upon the web it is clearly seen that the flanges

are nearly fully extended, while the web still preserves its original length, so that this method of rolling cannot do otherwise than produce a great strain in the rolled material, and very great wear and tear to the rolls. Further, this interior strain, produced as described above while rolling, remains in the finished girder. This is more clearly described in Mr. Sack's paper already quoted. In consequence, the ordinary rolled girders, as produced up to the present, are not reliable, as a proof of which I may mention that the web is liable to spring apart suddenly, if one chips away the flanges of a girder produced under the ordinary method. Another great disadvantage is, that at present the flanges must still have a certain amount of taper and are accordingly not parallel. All these important disadvantages are entirely obviated by the Sack universal girder-mill, with which one can produce girders with absolutely parallel flanges, of a thoroughly sound and naturally rolled material, which would be more suitable and reliable for any application than any other girder rolled up to now.

With the mill as described in Mr. York's paper I do not see how it is possible to produce girders with absolutely parallel flanges, as the Sack mill will produce, and I do not see how he prevents the formation of fins, or how he can arrange his mill to operate in any ordinary stand of housings, whereas Mr. Sack's mill can be adapted for an ordinary pair of housings, which can be used either for the ordinary or for the universal system of rolling girders—*i. e.*, with four rolls lying in the same plane.

I am prepared to show the feasibility of this to any gentleman present at the meeting, who may care to visit Mr. Sack's works, near Düsseldorf, where the old model of the year 1888 can be seen working, while I may further mention that Mr. Sack is at present building a universal girder rolling-mill after his system, which I expect will shortly be running.

Comparison of American and Foreign Rail-Specifications, With a Proposed Standard Specification to Cover American Rails Rolled for Export.

A Discussion of the paper of Albert Ladd Colby, p. 576.

E. WINDSOR RICHARDS, London, England:—In reading this paper the most interesting point to me was the question of the maximum percentage of phosphorus allowable in the steel rail. Mr. Colby said, and we will all agree with him, that the engineer knew, and even a steel-rail maker grants that phosphorus is the most undesirable constituent of steel. We in England have been for a long time considering a specification for the supply of steel rails. This matter has been taken up by the Institution of Civil Engineers, the Mechanical Engineers, the Iron and Steel Institute, the Institute of Naval Architects, and the Electrical Engineers. Committees have been formed, and the whole matter has received most careful attention. They arrived at last at an analysis which I will read: "The carbon is to be from 0.35 to 0.5 per cent.; the manganese from 0.7 to 1.0 per cent.; the silicon, not to exceed 0.10 per cent.; phosphorus, 0.07; and sulphur, 0.07 per cent." As to the phosphorus, which is the most important point of all in the analysis, we have for many years, and indeed until very lately, always agreed to supply steel which would not contain above 0.06 per cent. of phosphorus. Iron-ores are not quite as good now as they were formerly, and the manufacturers at the meetings of the committees referred to, tried to obtain an increase in the allowance of phosphorus. They asked the Sub-Committee to agree to 0.08 per cent. of phosphorus. They tried all they possibly could to obtain that; but they failed. English engineers in Great George Street and all over England, representing the important Associations I have mentioned, would only agree to 0.07 per cent. and that is the maximum allowance of phosphorus. Mr. Colby is an able advocate, but with all his ability he would not have been able to persuade the English engineers to agree to 0.08 per cent.

Mr. Colby very ably advocates that the Americans at any rate can agree to 0.10 or 0.11 per cent. of phosphorus. English engineers consider that that is a percentage much too high, and will not agree to it on any terms whatever. And so I think that we will have to be content with 0.07 per cent. on this side of the water. Mr. Colby tells us that it is impossible for American railmakers to produce rails under 0.10 per cent. Such rails would not be received in Great Britain. I do not think in the other points of the analysis there is very much difference between us. Mr. Colby gave the chemical composition (refer to page 612 of this volume of the *Transactions*), which, with the exception of phosphorus, I think our Standards Committee would agree to; but I think the question as to the amount of phosphorus allowable in a steel rail is the most important of all and perhaps the one which will be discussed more to-day than any other.

R. Price-Williams, London, England:—Mr. Colby very kindly sent me a copy of this interesting and valuable paper, which I should much like to see in the hands of some of the chief railway engineers in this country. I observe from the comparisons made in it of the American and foreign steel rail specifications, that the physical tests vary a great deal, and among them the most notable, and in my opinion the most objectionable, are some of the "drop-tests," which are extravagantly disproportionate to the force of impact a steel rail is ever subjected to on a railway, and the requirement of such needlessly severe tests necessarily involves a considerable reduction in the percentages of phosphorus and carbon, in order to secure the material from any risk of fracture from brittleness. With regard to the chemical tests, a long list of the results of which is given in the paper, a close agreement is observable in the percentages of the requisite constituents of the material for the manufacture of steel rails of the best quality, so far, at least, as regards strength, elasticity, and freedom from brittleness and risk of fracture. No reference, however, is made in this or indeed in any other paper on the subject I have seen of late years, to another important quality in a steel rail which, from the railway company's point of view, has of late years de-

served a great deal more attention than it has received—viz., its durability and capability of withstanding the destructive effects of the enormous development of railway traffic, more especially in the increased weight and speed of the trains, which has occurred during the 40 years which have elapsed since the first Bessemer steel rails were laid down in "the running roads" of British railways. It was, in fact, the much more durable quality of the material which, in spite of its then much higher price, so quickly led to its general adoption as the best and most durable material for the permanent-way of railways.

In this connection I should mention that in the early days of Bessemer steel manufacture, in which I was engaged, I gave a great deal of attention to the question of the durability of the material, and in the paper I read at the Institution of Civil Engineers I gave particulars of the actual amount of wear of the rail-heads of a number of steel rails laid down on portions of the Great Northern Railway, where they had been subjected to the destructive effects of the keenest traffic during a period of about eight years, the live and dead weight tonnage of which had been carefully ascertained by the then chief engineer, Mr. R. Johnson, M.I.C.E. A chemical analysis was also made of the constituents of some of the rails which showed the least and the maximum amount of wear, and portions of those rails were subsequently subjected to physical tests at Kirkaldys. The results given in the paper show that the rails which have suffered the least amount of wear of the rail-head are those containing a larger percentage of phosphorus than is now generally adopted as a maximum in British specifications, which, as Mr. Windsor Richards has just stated, is 0.07 per cent.; and coming as this does from him, with the weight of his great experience and authority as a manufacturer of steel rails, that percentage must be accepted as about the maximum which in this country is considered essential to insure the steel rail from any risk of fracture due to the brittleness of the material.

The maximum percentage of phosphorus, however, as given in the long tabulated list of American specifications, is considerably larger than is considered justifiable in the specifications in this country, and in many cases a maximum of 0.10 per cent. is specified. It is remarkable that the Great Northern

rail which in a period of nearly eight years showed the least amount of 0.12 in. of wear of the top table, after having been subject to a traffic of 59$\frac{2}{3}$ million tons, contained just that amount of 0.10 per cent. of phosphorus. It is also well worthy of note that it was a distinguished American engineer, Mr. Chanute, an honorary member of the Institution of Civil Engineers, who first drew attention to the greater endurance of the Great Northern steel rails alluded to which contained the higher percentages of phosphorus, which led to his adoption of the phosphor unit as a standard.

The maximum percentage of phosphorus observable in the American specifications is 0.10 per cent., as already stated, and the minimum 0.07, the exact maximum percentage adopted in this country. It would be interesting to know the amount of wear of the rail-heads or top tables, and of the traffic tonnage which had produced it, in the case of some of the American rails containing the maximum and the minimum percentages of phosphorus. The results, however, obtained from the Great Northern rail-tests sufficiently show that although, when devoid of any phosphorus, a steel rail is rendered less liable to the risk of fracture from an excess of it, the presence of some moderate percentage, as yet indefinite, certainly has the effect, as Mr. Chanute has pointed out, of very largely increasing the durable quality of the material.

To insure immunity from any risk of fracture is obviously the primary object of all steel-rail specifications. With the long and valuable experience we now have, its serviceable life as measured by time is now quite an easy matter. What is most needed now is the means, equally available, as in the case of the Great Northern rails, of ascertaining the serviceable life of these American rails as measured by the actual amount of wear of the rail-heads of those containing the maximum and the minimum percentages of phosphorus, together with the amount of the traffic tonnage which caused it.

There can be no question that the wear of steel rails subjected to the destructive effects of the great increase in the weight and speed of the main-line traffic of the principal railways in this country is far greater than is generally supposed, as is testified in fact by the results of the tests of the Great Northern rails, already alluded to, where in some cases as much as

0.5 in. of the rail-heads was worn away in the short space of about eight years (practically the serviceable life of the rail as measured by the traffic tonnage and amount of wear of the rail-head), while other rails almost adjoining them, and subject to the like amount of traffic tonnage, and possessing all the other requisite qualities to insure safety from fracture of the material, experienced only one-fourth the amount of wear. Under these circumstances it is a matter of great importance from the railway company's point of view that only steel rails possessing the highest qualities of durability consistent with those for insuring security from risk of the fracture of the rail, should be used, and thus maximum serviceable life obtained.

There is everything to indicate that with some slight modification of the percentages of the constituents of the material, a rail of at least as high or even a higher quality of durability than that attained in the case of the Great Northern may soon be regarded as an essential requirement in all steel-rail specifications.

The annual cost of the maintenance and renewal of the permanent-way of the principal railway systems in Great Britain constitutes a large item of the working expenditure; and taking the London & North-Western Railway, the premier railway, by way of illustration, it amounted in 1904 to considerably more than half a million, or just one-seventh of the entire working expenditure. It is scarcely necessary to say, that anything like an approach to an increased durability of steel-rail material, such as mentioned in the case of the Great Northern rails, would largely contribute to the reduction of the working expenditure of that and most of the other great railway systems, which has now reached the exceptionally high figure of from 63 to 64 per cent. of their gross traffic receipts.

F. W. Harbord, London, England:—As to this question of phosphorus, we all agree that a specification should be drawn so that manufacturers can conform to it, and, if the position in respect to ores in America is such that 0.1 per cent. of phosphorus is the minimum that they can give in their rail-steel, it is only reasonable that this limitation should be made. In England I think we may say that the manufacturers are in a rather happier position, inasmuch as they can work regularly

and systematically without any great trouble to a specification of 0.08 per cent. of phosphorus, and if they can do this, there is no reason why the limit should be raised to 0.10 per cent. I think 0.08 per cent. is a more reasonable limit than 0.07 per cent., but still the powers that be have settled upon 0.07 per cent. and I think we can, by taking great care, work to that. I take it that this 0.10 per cent. phosphorus limit refers to the Bessemer process, but now that the basic open-hearth process is coming largely into use in America, there should be no difficulty in making rails 0.08 per cent., or less if required. The question of manufacture has an important bearing upon the carbon, and I think that it is a great pity that the content of carbon in reference to manufacture has not been dealt with in specifications. My experience is that with a given content of carbon we get a different degree of hardness, depending largely upon the method of manufacture—that is to say, a basic open-hearth rail of 0.50 per cent. of carbon is distinctly softer than a steel rail made by the acid process. So that for the basic open-hearth rail we can take a higher carbon with lower phosphorus, and still be within the region of safety. I have had lately some rails brought under my notice which gave most excellent drop-test results, containing from 0.6 to 0.7 per cent. carbon, made by the basic open-hearth process. This question of the influence of manufacture will have to be considered, as otherwise if we have the same carbon-content for basic open-hearth as we have been accustomed to specify for acid steel, we will get the rail-heads spreading, and other troubles. We must not, therefore, draw hard-and-fast lines by saying that we will not have rails of a certain percentage of carbon without taking into consideration the method of manufacture.

R. A. HADFIELD, London, England:—I wish to compliment Mr. Colby for the information he has given to us. We wanted to know what our American friends were doing; but I think if Mr. Colby had made his propositions to the Standards Specifications Committee he would have found a large number of English engineers against him, and he would have found difficulty in persuading them that steel containing 0.10 per cent. of phosphorus is a safe material to be used. Mr. Colby gives us a table of American rails not having been broken, although

they contained 0.10 per cent. of phosphorus. I would like to ask Mr. Colby as to the remaining constituents present. It is quite possible to have 0.10 per cent. of phosphorus, provided that the carbon is not too high, and the manganese is sufficiently high. M. Euverte, many years ago, read a paper in reference to the combined influence of manganese and phosphorus upon steel, in which he showed that phosphorus was not so deleterious as was thought; at any rate, that was his opinion. Mr. Colby stated that if the carbon was low we might push up the percentage of phosphorus probably to the limit which was mentioned; but, as he also pointed out, in American conditions, the higher the carbon, within certain limits, naturally the greater the durability of the rail. I do not know whether Mr. Colby was present at the National Physical Laboratory on Tuesday, where we had the alternating-stress machinery at work. I would suggest to him that if he could send some specimens containing as much phosphorus as he spoke of, and let Dr. Glazebrook carry out the tests upon steel containing high phosphorus, he would then find out whether it is a safe material when the carbon is at the same time high. The paper is an excellent one. It has elucidated points which have been under consideration for the last few years, and we are very much indebted to Mr. Colby for it.

J. E. Stead, Middlesbrough, England:—I have only one or two remarks to make in reference to the standard of phosphorus in rails. I think chemists and metallurgists in this country are all agreed that if they raise the carbon, the effect of phosphorus becomes more and more pronounced. I cannot go into the reasons for it on this occasion, but they are pretty well known scientifically. If we raise the carbon we must lower the phosphorus. With reference to the standards instituting 0.10 per cent. of phosphorus, I think it would be a mistake to make it so high in this country and for the United States of America, for the reason that if we allow 0.10 per cent. in a contract, manufacturers naturally will say there should be a swing of the pendulum with 0.10 per cent. as about the average, and they should be allowed one or two points above 0.10 per cent. For this reason it is proper and correct to place the basis low, so that the average is not more than 0.07 or 0.08 per cent.

But I think we would be very ill-advised to reject rails if the limit exceeded in one particular rail, or a few rails, 0.07 or 0.08 per cent. provided the other elements are not very high or in objectionable proportion; then 0.10 per cent. even might be allowed without rejection. I find that as regards broken rails, high manganese causes more fractures than phosphorus. High manganese and high carbon together are very treacherous, and while high manganese makes the rails brittle if they were rolled and cooled on a cold winter's day, yet, if rolled in the summer, and the rate of cooling retarded, they would most probably be all right, and wear better than rails of normal composition. Manganese should not exceed 1 per cent. in rails. I think that metallurgists do not pay sufficient attention to the effect of manganese in rails, and sometimes the brittleness is put down to the phosphorus instead of to the high manganese.

JAMES E. YORK, New York, N. Y.:—As a boy I was associated with the rolling of double-headed rails. I think the bull-headed rail had not been introduced at that time. The physical treatment of rails has a great deal to do with their durability. Mr. Stead has said that on a cold day high manganese is rather destructive to the tenacity of the rail. Mr. Stead might also have said that a low finishing temperature in rolling the rail is also detrimental to its physical qualities from the fact that the section of the rail does not permit of a uniform flow of metal at the same surface speed per minute throughout the section. That condition is apt to leave an internal stress in the rails, which may thus yield to a sudden blow in the track. That applies in a much greater degree to the rolling of a T or Vignoles rail, as used in America and elsewhere, than it does to a double-headed or bull-headed rail as used in England, from the fact that in the double-headed or bull-headed rail both the base and the head are generally of the same width, thus permitting a more uniform flow of metal in the rolling. That, in my opinion, accounts for the less frequent fractures in the double-headed or bull-headed rail than in the T-rail. I wish to ask why it is that rails in Great Britain do not break to such a degree as the T-rails of the United States of America. In my opinion it is entirely owing to the difference in the shape of the two rails. If the rails are rolled at

a low temperature, so as to get the best physical results through securing the fining of the grain, the result is an unnatural stress left to a greater extent in a T-rail than in a double-headed rail, because of the difference of the uniform flow of metal during rolling. To illustrate that, I would point out that the diameter of the roll to form the web of a T-rail is in some instances at least 5.5 in. larger than the part of a roll which forms the extreme width of the flange. The result is that the small diameter only gives off the metal at a much slower rate than the part forming the web, consequently the wider part of the section will either slip or stretch during the operation, and the result is that internal stresses are left inherent in the finished section. That occurs much more with the T-rail than it does with the double-headed rail referred to, which lends itself to a more uniform flow of metal than the T-rail, during rolling. An illustration of that can be seen by the much larger amount of crop-ends from a T-rail when rolled, than from a double-headed or bull-headed rail, the conditions as to ingot or billet being the same. Mr. Hunt some time ago in one of his papers stated that some of the rails made in Great Britain in the early sixties had proved conclusively that the heat and physical treatment were much more important than the chemical constituents. They were in the track in active service 35 to 40 years, and their durability had been so satisfactory that it was decided to have them analyzed, and they were then found to contain three times as much phosphorus as is now thought judicious, and also other impurities. In spite of that they had not broken, and answered the purpose for which they were produced. Now railroad engineers are complaining of the poor quality of the rails produced by modern practice. Railroad engineers have asked me what the cause of the difference in quality between the rails made in the past and in the present is, but, as I am not interested in the manufacture of rails, I said very little about it. In my opinion, however, the good quality of the older rails has been largely due to the mechanical treatment of the metal at the time of rolling.

A. Lamberton, Sheffield, England:—I wish to refer to a point not touched on by the previous speakers. A good deal has been said as to the percentage of phosphorus permis-

sible in rails, and Mr. Colby has referred to phosphorus as high as 0.11 per cent. in some American rails, which had given good results; but that would be regarded as quite unsafe in this country. I think the probable explanation is to be found in the difference in construction between American railroads and those in this country. In America the rails are invariably of flat-bottom section, resting directly on wood ties placed at closer centers than in British practice. This form of construction is beneficial in that it tends to modify the intensity of the shock and vibration imparted to the rail. I think that a rail with a tendency to brittleness would be likely to stand better under the conditions due to this form of construction than if it formed part of a British railroad, where the rails rest on hard cast-iron chairs and the shock and vibration are more severe. Certainly rails having 0.11 per cent. of phosphorus would never be accepted in Britain, and I think it too high, no matter what form is adopted. At the same time, I believe it might be less objectionable in the American system of construction than in the British.

ROBERT W. HUNT, Chicago, Ill.:—I am convinced that the process of manufacture in America will be forced to change owing to ore conditions, and I believe that in a very few years the question of limitation of phosphorus will lose its significance in the country. At the present time, however, it exists. There is a great deal of difference of opinion about it. A great organization, the Engineering and Maintenance of Way Association, composed practically of all the representatives of the railroads of the country, has adopted a specification in which they limit the phosphorus to less than is proposed in the paper. It is not fair, however, to say that the American Society of Civil Engineers have committed themselves to that position, because as the report of their Special Committee on Rails has not yet been accepted by that Society, its position cannot be assumed.

E. F. KENNEY,* Philadelphia, Pa. (communication to the Secretary†):—The author has certainly been misinformed re-

* Engineer of Tests of the Pennsylvania Railroad Company.

† Received July 13, 1906.

garding the rail situation on American railroads. Very convincing testimony has been furnished to both the American Society of Civil Engineers and the American Railway Engineering and Maintenance of Way Association, showing that nearly every American railroad having heavy traffic is suffering greatly from broken rails. The rails are not giving satisfaction as to wear; and any attempt to improve the wear by making the rails harder is met by a crop of brittle rails. This brittleness is caused by high phosphorus. Rails with lower phosphorus have been made, in which carbon was quite high, thereby getting much better wearing-qualities without being brittle; but as long as the phosphorus is kept up around 0.1 per cent. it will be impossible to get rails which will wear well without being dangerous. Far from being what the author seems to think it, the brittle rail is one of the most important subjects before American railroads to-day. Phosphorus is an unmixed evil; and any reduction in phosphorus-content will mean much to the users of the rail, both as to wear and safety. If the softness asked for in the foreign rails is required for safety, raising the phosphorus-content will necessitate lowering the carbon, to get equal insurance against brittleness. The arguments in favor of raising the allowable phosphorus-content are not sound. Careful heating and lower finishing-temperature might overcome a difference of 0.02 in phosphorus if the lower-phosphorus rail had been heated too high and finished too hot; but there is no reason why the low-phosphorus should not be as well heated and finished as the high-phosphorus material.

Moreover, there is small likelihood of getting rails well-heated and finished under the proposed specifications. They contain no limitation whatever of the finishing-temperature, so that cold-finishing is not likely to be obtained. The recent American specifications all fix a maximum shrinkage allowance to regulate this, but it is steadily opposed by the manufacturers.

The proposed specifications make no mention of hot-straightening. This is most unfortunate, since many of the broken rails reported by American roads can be traced directly to injuries received in straightening. The hot-straightening is often carelessly done. The rails come down to the straighteners in very bad shape, which necessitates a great deal of severe gagging in the straightening-presses. This evil has attained

such proportions that I have proposed the specification of a maximum camber for rails arriving at the straightening-presses, and the requirement that all rails having a greater camber, or having sharp kinks, be marked as No. 2 in quality before being straightened, and only accepted as such. This requirement has been accepted by the American Society of Civil Engineers Committee on Rails, and incorporated in their specifications. It has also been adopted by the American Railway Engineering and Maintenance of Way Association, and put in their specifications. A little more attention paid to hot-straightening will reduce greatly the number of rails ruined in the gag-presses.

The word "sufficient" in Clause (*e*), relating to the cropping, should be qualified to make it more definite. It has been used in most American specifications until recently, and has been uniformly interpreted by the makers to mean "sufficient from their point of view." The inspector at the mill has no voice in the matter. All that is now done in shearing is to cut away the material until no black spots are shown when the bloom is cut through. There is no attempt to remove the zone of greatest segregation and unsoundness. This is not sufficient to insure sound material, and makers know it; yet they oppose the specification of a definite discard. A certain percentage of the whole ingot should be specified as the minimum discard from the top.

W. E. Freir, London, England (communication to the Secretary*):—The presentation of Mr. Colby's paper affords an opportunity for bringing forward a matter which is of very great importance, and which is causing much anxiety in almost every town of the United Kingdom and the Continent of Europe, as well as in many American cities. I refer to what is termed the "corrugation" of tramway-rails. In the French-speaking countries these corrugations are styled "*les ondulatoires,*" and the Germans refer to "wave-like wear." By whatever name they are known, however, these corrugations represent not merely a nuisance to the tramway engineer or manager, but also a source of monetary loss, which is of the greatest moment to the corporations and companies owning or working the tram-

* Received July 14, 1906.

ways. The cause of these corrugations is at present shrouded in mystery, and although any number of theories have been put forward, not one of them seems to be able to withstand serious practical tests. The subject was first brought to notice in *The Light Railway and Tramway Journal* in December, 1903, and January and February, 1904, when various theories were propounded as to the cause, including contributions by Mr. J. E. Stead, and several American engineers. In 1904 the question was discussed in Germany by Von Borries, Scheibe, Schwarbach, and others, and it was also found that under the guise of "roaring" rails, the defect was quite rampant on many of the lines of railway in India. More recently the subject has again been taken up very fully by *The Light Railway and Tramway Journal*, and there can be no question that it deserves the attention of rail manufacturers generally, seeing that, to the tramways of the world, it involves the possible expenditure of many millions of money for the replacement of faulty rails, and that without any guarantee that the mischief may not recur on the new rails.

The theories which have been put forward, as to the cause of these corrugations, are so numerous that they cannot very well be enumerated, but that which is of most interest to metallurgists, and particularly to rail manufacturers, is that which attributes the trouble to faults in manufacture, arising either through segregation in the ingots, improper rolling, or an incorrect finishing-temperature when rolling. Certain contributions read before this Institute, and other technical bodies, lend some color to the idea that the fault may be originated during the process of manufacture, but, on the other hand, there are many facts which lead to the conclusion that the corrugations are in no sense due to the composition of the steel or its manipulation. Then there are those who attribute the mischief to the insufficiency of the carbon, or to too-great percentages of manganese. The average weight of tramway-rails (girder section) in this country is 90 to 100 lb., and the general specification calls for carbon, 0.35 to 0.5; manganese, 0.7 to 1.0; silicon, not over 0.10 (generally 0.07 or 0.08); phosphorus, not over 0.07; and sulphur, 0.07 per cent. This, it will be seen, gives a fairly soft rail—softer at all events than the street-railway rails (T-section) mostly used in the United States; but there

would not seem to be anything in the analysis which should lead to the trouble under discussion. Nor is it easy to attribute the fault to the method of laying the rails. The theory that the use of the rails as the return half of the electrical circuit causes the corrugations, is disposed of by the fact that corrugations exist equally on cable- and steam-tramways, where no current is used. In the same way the hypothesis that, as almost all tramway-rails are made of Bessemer basic steel, therefore the process is responsible, is also dismissed by the existence of the faults on rails made of Bessemer acid steel. Tramway girder-rails in the United Kingdom are laid on a bed of concrete, and anchored down in the most rigid manner possible, fished and bonded at the joints, with no allowance whatever for expansion—contrary to the suspended flexible joints of railway practice. Yet the "roaring" rails appear in India, on the ordinary railways, and corrugations show in all sorts of places, on up-grades and on down-grades, on straight line and on curves, and on electric railways with bull-headed rails, but only, in this instance, where check-rails are used. The literature of the subject is rather meager, being practically confined, in this country, to *The Light Railway and Tramway Journal*, and in Germany to *Glaser's Annalen* of 1904, yet there can be no question of the high importance of the matter, and that it is one which merits the attention of the Iron and Steel Institute and of the American Institute of Mining Engineers. Tramway engineers are endeavoring to remedy the corrugations by grinding the head of the rail with emery or other grinders, but this is obviously a mere make-shift arrangement, and they would welcome any suggestions which would prevent the occurrence and recurrence of the fault. Is it possible that a harder rail would be less liable to develop these corrugations? Do American rails of 90 to 100 lb., with 0.50 to 0.60 per cent. of carbon, and 0.80 to 1.10 per cent. of manganese, and phosphorus up to 0.10 per cent., wear free from these wave-like depressions? Is there anything in the idea that the rigidity of the tramway-track is conducive to the production of the mischief?

These are questions which are of moment, and if the discussion on this paper should furnish answers to them, I am sure the tramway-men of the world would be very grateful for the information.

WILLIAM R. WEBSTER, Philadelphia, Pa. (communication to the Secretary*):—In this country rails for export-orders have been and are now being manufactured and tested in strict accordance with all of the requirements of some of the most severe specifications referred to by Mr. Colby. It is only after an engineer has been convinced that rails can be made here equal in every respect to those he is getting abroad, that he should be asked to modify any conditions in his specifications, and then only to accept such requirements as have the indorsement of our railway companies and engineers.

The standardization of rail-specifications in this country is in very good hands: the American Society of Civil Engineers, the American Railway Engineering and Maintenance of Way Association and the American Society for Testing Materials have each appointed special committees to do this work. Some of the members of these committees serve on two committees and some on all three. Much of the work has been done independently, but the findings of each committee have been considered by the others, and joint meetings have been held. But notwithstanding all this, there are still differences in their specifications on the following points, which are now under consideration by all the Societies and committees:

Chemical Composition.—One of the committees calls for lower phosphorus and higher carbon than are usually specified. The second agrees with this, except that they have a note stating that the carbon can be modified to suit local conditions. The third calls for higher phosphorus and lower carbon than the first, claiming that equivalent results can be obtained in the rails if they are rolled properly.

Drop Test.—All of the committees agree that the butt end of rail taken for test shall be from the top of the ingot. Two of them agree on the height of drop for each weight of rail, and require a test from each heat of steel; the other committee considers one test from every fifth heat of steel, and a less height of drop, sufficient, on account of rail being taken from the top of ingot.

Discard.—Only one of the committees considers the usual clause—"sufficient discard from the bloom shall be made to insure sound rails"—is satisfactory. The second asks for more

* Received July 13, 1906.

than the usual amount and specifies the length from bloom to be discarded. The third does not consider this satisfactory and specifies a given percentage of the weight of ingot to be discarded, in order to cover the use of different weights of ingots.

Temperature of Rolling—Shrinkage Clause.—All of the committees appreciate the importance of putting enough work on the steel at a low temperature to break up the coarse structure. They agree that the temperature of the rail at the time of leaving the finishing-pass is a fair indication of how this work has been done, the other conditions of rolling being known. Also, that the amount a rail of given length contracts or shrinks after being cut at the hot saws, in cooling to normal temperature, is the most convenient check on the finishing-temperature. As the distance of the hot saws from the rolls varies at the different mills, the time between the rail leaving the finishing-pass and its being sawn also varies: a correction has to be made for this time, and has been agreed on by all three committees. They also agree on the differences in shrinkage called for between the heavier and the lighter rails. Two of them agree on the amount of shrinkage to specify at the time of the rail leaving the finishing-pass and the other asks for a larger amount, or higher finishing-temperature.

As a member of the committee of the American Society of Civil Engineers, and as Chairman of the other two committees, I desire to state that an earnest effort is being made to harmonize these differences, and to secure rails best suited to withstand the severe conditions imposed by increase of wheel-loads and speed of trains.

It is expected that standard specifications will be agreed on that will be satisfactory to all.

The Standards Committee of England has, through its Secretary, been kept advised of what is being done in this country: and the Secretary, in his report on the visit to the United States in 1904, refers to the work as follows:

"In regard to the unification of specifications between the two countries, there is no doubt that a closer co-operation between our Committee and the American Society for Testing Materials would lead to a harmonizing of methods of testing, and where the practice permitted it, of specifications. The essential differences in practice existing in the two countries will prevent any complete harmonization of sections and specifications; but there are many points upon which co-operation might be

secured, to the mutual advantage of both countries. This could be assisted by more complete interchange of views between the Engineering Standards Committee and the American Society."

C. S. R. Palmer, London, England (communication to the Secretary*):—The paper contributed by Mr. Colby I understand to be written for the purpose, not of pointing out what is good or bad in the analysis and manufacture of rails, but of drawing attention to points in specifications he has seen which cause needlessly extra expense in manufacture, such expense falling ultimately on the clients of the drawer of the specifications.

It is unquestionable, of course, that some of the safeguards he draws attention to are now unnecessary, because of the possibility of attaining greater certainty in the methods of manufacture. But this advance in certainty of method has been accompanied by the possibility of much greater rapidity of manufacture than previously obtained, with the net result in some cases that enough work, physical and mental, is not put into the rails; and, moreover, there is a third possibility—viz., of work being slurred over, owing to the checks or tests being few.

In this connection, it has to be remembered that a reduction of price on the rails of 10 per cent. at the mills will work out to probably not more than 6 per cent. on the rails laid in the road abroad, and probably not over 2 per cent. of the whole cost of the very cheapest of railways. This small saving would be dearly bought if the life of the resulting permanent-way were at all reduced.

The matter, however, is one of considerable interest to me, and attention is therefore drawn briefly to some main points in Mr. Colby's proposals which would tend to prevent the competition he presses for, since these points, as well as sundry others, either give the manufacturer undesirable latitude or curtail the rights of the purchaser and his engineer to an extent that it would not pay the purchaser to accept.

Taking first that most objectionable element in steel—viz., phosphorus—Mr. Colby urges acceptance of a percentage of 0.10, against 0.07 usually allowed in Europe, and at the same time he also urges reduction of the amount of drop-testing

* Received July 21, 1906.

to one-half of that considered necessary in Europe and by some American railways. It might be possible as a matter of economy to ease one or the other, but surely not both conditions; that is to say, it might be possible to accept the higher phosphorus, but, in my opinion, only when accompanied by a rigid drop-test on a piece from the top of the ingot of every heat—the top, that is, after the specified proportion of the ingot has been discarded. Moreover, acceptance of the higher phosphorus with the carbon-percentage (higher and not lower than Mr. Colby's in the smaller sections) allowed in England would make for the necessity of greater rigidity in adherence to analyses and of not less frequent analyses.

Further, in connection with the quality of the rail, is the question of the finishing-temperature and the amount of work therefore put into the rail. This does not appear to have been touched on in the paper anywhere, and I would be glad of the author's remarks on the point. Assuming too that the temperature is sufficiently low for good work, is not an allowance of $\frac{3}{16}$ in. each way in a 33-ft. rail sufficient for hot sawing?

Moreover, from the point of view of the purchaser seeking economy, there is an objection to his being called on to pay for a greater weight than he contracted to buy. Thus, while it is reasonable to give the manufacturer a rolling-margin both ways in weight, it is hardly right and certainly not economical for the purchaser to take the risk of having to pay for overweight, more especially remembering that 0.5 per cent. excess of cost at the mill means from 0.75 to 1 per cent. excess of cost in the road abroad. Similarly, as regards accepting short rails as part of the tonnage contracted for, it should be remembered that the amount of short rails recommended by the author would mean 1 per cent. extra joints in the line, with, of course, the corresponding extra capital-cost of jointing, and of sleepers, and of maintenance. Yet again, why should the purchaser be called on to accept seconds in addition to the tonnage contracted for? Will it pay him to be left in doubt as to what quantity the manufacturer's methods of work, good or bad fortune, may cause him to be billed for?

Finally, as illustrating a third class of objections, attention is drawn to the proposal to curtail the engineer's powers, and, at the same time, to enter in the specifications such general word-

ing as "The entire process of manufacture and testing shall be in accordance with the best current practice," and "Sufficient material shall be discarded or cropped from the top of all ingots to insure sound rails." There should be some judge in case of dispute between manufacturer and inspector; the purchaser cannot afford to take the risk of expensive arbitrations; hence the English practice of vesting final decision in the specifying engineer, and, so far as my experience gained in three continents goes, the practice does not bear hardly on, or alternatively in favor of, either purchaser or vendor.

ALBERT SAUVEUR, Cambridge, Mass. (communication to the Secretary*):—The most glaring shortcoming of the rail-specifications reviewed by Mr. Colby as well as of those which he proposes is to be found in their silence on the subject of "discard."

It has become customary in American technical meetings to dismiss this question of discard by some more or less witty jest, but such practice only serves to illustrate the weakness of the position of those resorting to it.

I have repeatedly called the attention of manufacturers as well as of consumers to the very serious danger of rolling "piped" rails, which is the common practice of the day. The pipe extends so far down the ingot that the first rail rolled must, in the majority of cases, be "piped" and, therefore, seriously defective, because of insufficient discard. This fact is, of course, well known to manufacturers, and their fear to have it brought to light in a manner too costly to themselves accounts for their strenuous and so far successful opposition to the introduction in specifications of a clause requiring the drop-test to be applied to the rails corresponding to the tops or piped ends of the ingots. It is not to be supposed that the consumers are not equally well informed regarding the existence of this defect and their failure to insist upon such specifications as would prevent the rolling of piped rails is more difficult of explanation unless it be assigned to their fear of having to pay considerably more for their rails in case of a much larger discard. In the United States, moreover, the financial relations between large railroad companies and steel-works are often so close as

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* Received August 31, 1906.

to suggest at least another explanation for their apparent indifference.

So long as so vital a matter is ignored in rail-specifications, I, for one, am unable to take much interest in the refinements suggested in chemical composition, finishing-temperatures, etc., —they certainly appear futile, not to say farcical. It is like trying by ingenious devices to improve the complexion of a patient suffering from a severe organic trouble. Is the game worth the candle? What is needed to save him is a major surgical operation and not the administration of homeopathic and relatively ineffective pills. Continued failure in this respect must result in placing these makers of rail-specifications in the metallurgical branch of a great family of unsavory fame.

M. Nigond,* Paris, France (communication to the Secretary†):—The chief difference between the specifications adopted by the company with which I am associated, and those suggested in Mr. Colby's paper, is that my company do not require a specific chemical composition, but, on the other hand, they do specify the tensile strength, with a clause as to the minimum elongation, and also drop-tests and bending-tests.

They consider that chemical determinations, except carbon, cannot be furnished with sufficient accuracy, without retarding manufacturing operations, and it therefore appears safer to judge the metal by the actual results it gives, rather than by its composition. Slight variations in that composition may, as a matter of fact, have an important influence on the strength.

In the case of their rails of 42 kg. per m. (85 lb. per yd.), when it was specified that the test-specimen for tensile should be taken from the portion of the head least liable to wear the required tensile strength was reduced from 70 to 67 kg. per sq. mm. (99,562 to 95,295 lb. per sq. in.) and the minimum elongation from 10 to 8 per cent.

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* The Engineer in Chief of the Paris and Orleans Railroad.

† Received July 20, 1906.

The Gas-Producer as an Auxiliary in Iron Blast-Furnace Practice.

A Discussion of the Paper of R. H. Lee, p. 366.

J. T. PULLON, Rowangarth, Roundhay, Leeds, England:—In discussing Mr. Lee's paper, I wish to call attention to the fact that Mr. B. H. Thwaite (who was heard here yesterday on the subject of the application of blast-furnace gas for the production of power, of which he was undoubtedly the pioneer) read, at the Engineering Congress in Glasgow in 1901, a paper on the profitable utilization of power from blast-furnace gases,[1] in which he suggested the diversion and cleaning of the whole of the waste gases coming from the blast-furnace and their utilization for power-production, so as to obtain, with proper manipulation, from 4 to 6 times the efficiency of present methods. He suggested also the heating of the stoves with producer-gas of a higher and therefore more suitable calorific value, so as to maintain them in a constant state of maximum efficiency, free from dust, and to avoid the irregular working of the furnace, besides obtaining a maximum supply of air-blast, of 15 or even 20 lb. pressure, by means of internal-combustion blast-engines driven by the cleaned waste furnace-gases, with a surplus of gas left for other uses.

Since that time, in view of the necessity of having a stand-by plant, immediately available, in case of strikes or other reasons causing the banking or blowing-out of the blast-furnace, he has developed, as an addition to his other types of producer, a high blast-pressure gas-generator, producing a gas identical with, or somewhat superior to, blast-furnace gas; and in which all the ash of the fuel is turned into fluid slag, which is available for the production of slag-wool. The gas is a little richer in carbon monoxide than average blast-furnace waste gas, and has only from 1 to 3 per cent. of hydrogen.

Fig. 1, drawn from a photograph, shows a plant containing this generator, now in operation at Leeds. It is made in units of from 1,000 to 10,000 h.p. capacity for each vessel, and coupled,

[1] *Journal of the Iron and Steel Institute*, vol. lx., No. 2, pp. 149 to 184 (1901).

of course, to Thwaite's ordinary gas-cleaning plant. It can be put in full blast in 2 or 3 hr., producing gas equal to blast-furnace gas, the most perfect of all power-gases.

The cost of the Thwaite plant is low, owing to the high pressure employed. It is automatic in its action, and the thermal efficiency is as high as practicable. There is no loss of un-

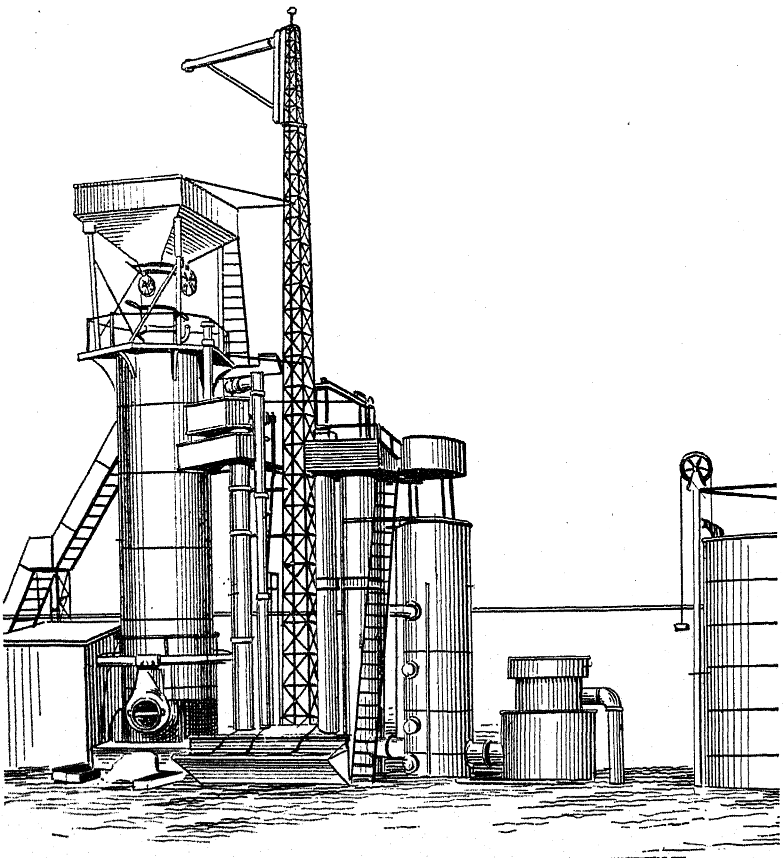

FIG. 1.—GAS-PRODUCER PLANT AT LEEDS.

burnt fuel passing out with the ash and clinker, a serious defect in other producers, especially important with those working at low temperatures. With an ordinary type of gas-engine of the comparatively small power of 100 h.p., a thermal efficiency equivalent to 8,000 B.t.u. per i.h.p. per hour has been obtained. I append two analyses of this gas, as made from ordinary gas-works coke, which contained 9.5 per cent. of ash.

Analyses of Generator-Gas.

	I. Per Cent.	II. Per Cent.
CO_2,	1.2	0.8
O,	0.4	1.8
CO,	30.0 (112.1 B.t.u.)	32.4 (110 B.t.u.)
CH_4,	0.2	0.4
H,	4.1	1.6
N,	64.1	63.0
	100.0	100.0

F. T. Havard, Silberhütte, Anhalt, Germany:—The utilization of waste gases from reverberatory furnaces (such as those used in the manufacture of glass, in the reduction of minerals —for instance, heavy spar—and for similar purposes) by the attachment of a producer, is already emerging from the experimental stage in Germany. It is perhaps worthy of mention that I am planning to make use of the cleaned gases from lead and copper blast-furnaces for the purpose of generating power by enriching and controlling the quality through the addition of comparatively small quantities of producer-gas of high calorific content, whereby the realization of the heat-values in these gases would be effected. I cannot yet tell how these plans will succeed on a large scale, but I could not miss this opportunity to tell what the copper- and lead-smelting men are doing in utilizing the waste gases by the help of the auxiliary gas-producer.

Prof. William Kent, Syracuse, N. Y. (communication to the Secretary*):—There are two sentences in Mr. Lee's paper which should not be allowed to pass without comment. These are:

(1) "The net calorific effect of coal burnt under the boilers and in the gas-producers, respectively, is, if not the same, rather in favor of the producer; indeed, the efficiency of carbon burnt in the form of producer-gas is claimed to be from 5 to 25 per cent. greater than that of direct combustion of solid fuel."

(2) ". . . gas-producers would have the advantage over the present mode of coal firing, that a greater calorific effect is obtained from the fuel."

Solid fuel burned under boilers under proper conditions of air-supply and of heating-surface may give an efficiency of 75 per cent. or more, the remainder or loss being accounted for by heat carried away in the chimney-gases, radiation, etc. There is no way by which the use of producer-gas can diminish these chimney- and radiation-losses. In fact, the furnace under

* Received Oct. 11, 1906.

a steam-boiler is nothing but a gas-producer and a combustion-chamber combined, and nothing can be gained by separating these two parts and connecting them by a long gas-flue or main. Since the invention of the Siemens gas-producer, 50 years ago, many producers have been made to be used in connection with steam-boilers, but they have always failed in practice to show any advantage over the use of solid coal. If gas-producers have not succeeded in displacing direct firing for ordinary boilers, it is not likely they will displace coal-firing under blast-furnace boilers.

If it is necessary in blast-furnace practice to have the means of supplying a large additional quantity of heat to the steam-boilers in the case of occasional stoppage or diminution of the supply of gas from the blast-furnace, this can best be accomplished by having a very large grate-surface under the front end of the boilers and a very large combustion-chamber above the grate, for the burning both of the gases from the blast-furnace and of the gases distilled from the coal. When the gas-supply is sufficient, the ash-pit doors may be shut and the coal will not burn. When the gas-supply is diminished in quantity the ash-pit doors may be opened and the coal burned by natural draft; or if the gas is entirely shut off from one or more boilers the ash-pit doors can again be closed and the coal burned by forced draft. With a grate-surface 8 ft. long and the width of the boiler-setting in water-tube boilers, and forced draft, bituminous coal could easily be burned at the rate of 40 lb. per sq. ft. of grate per hour. This would drive the boiler from 50 to 100 per cent. above its normal rated capacity.

To provide this amount of coal-burning capacity in gas-producers would require a very expensive producer-plant without any gain in economy over the coal-firing. A good plan for a water-tube boiler-setting for blast-furnaces is the one designed by Mr. Julian Kennedy for the Lucy blast-furnace.[2]

For the purpose of increasing the grate-surface under this boiler, I would modify that design by moving the gas-flue, gas-burner and front wall of the boiler-setting and the perforated wall outward 4 ft. in front of the present position, forming thus a gas-combustion chamber in front ot the boiler-setting, which would leave room for a grate-surface nearly 8 ft. wide, thus doubling the coal-burning capacity of the grate.

[2] *Trans.*, xiii., 46 (1884–85).

Gas-Engine Practice.

A discussion of the Papers of Prof. H. Hubert, p. 647; K. Reinhardt, p. 669; and Tom Westgarth, p. 796.

ADOLPH GREINER, Seraing, Belgium:—I have nothing special to add to Professor Hubert's paper except to say that there are some little things that it would be well to have corrected when the paper comes to be published. In dealing with the thermal efficiency of the engine, you will find that the 29.84 per cent. was a very high one. Professor Hubert has forgotten those engines built under the Cockerill type, but he has referred to them in Appendix II. At our works at Cockerill we have built 63 engines, giving 63,000 h.p., and the other companies who have received licenses have built 113 engines, making 103,000 h.p., and aggregating 176 engines and 166,000 h.p. Eight years ago I had the pleasure of giving to the Institute the results obtained up to that time. Since then a good many engines have been constructed. In 1898 I thought a blast-furnace producing 100 tons of pig iron would allow 3,000 h.p. Professor Witz and Professor Hubert show that with the same production of pig the new engines take 3,800 h.p. out of the gas. At present at Cockerill we make 700 to 800 tons of pig iron per day, and we hope to have 26,000 h.p. in a few years. At present we have only half of that in gas-engines, but we hope to have in five or six years all the gas out of these furnaces to the number of 25,000 or 26,000 h.p.

TOM WESTGARTH, Middlesbrough, England:—My contribution is not a paper in the ordinary sense of the word, because I am sure you will agree with me that what I have written is only supplementary to the other two papers. The notes are schedules of the larger-size gas-engines built by British engine-makers, of 500 h.p. and upwards. If you look at the schedules you will find that in England gas-engines have become quite ordinary motors for a good many purposes. It was at first thought that gas-engines were troublesome, and ill-

adapted for general purposes, that they were not of much use; but you will see now from the schedules that they are being employed for driving dynamos, working blowing-engines, and driving rolling-mills, tube-mills, air-compressors, cement-mills, paper-mills, electrolytic work, and even cotton-mill work, mine-fans, and so on. I do not think that it needs any argument to show that the gas-engine has come to stay, and that it is doing a great variety of very useful work. Another point in the schedules to which I would like to call your attention is the different kinds of gas being used. The primary object of this type of gas-engine is to use gas which hitherto has been wasted, but which a few years ago suddenly attained great value, because of the way in which it could be utilized with advantage in running gas-engines. In addition to blast-furnace gases, coke-oven gas, and various kinds of producer-gas, even that from bituminous coal has been used to work gas-engines of large size. I am sorry that when I asked Messrs. Beardmore for the particulars of their engines they did not send any particulars of their vertical marine-engines, because although that point is not particularly interesting to us as iron and steel makers, it is interesting to engineers, and has an important bearing on the question of the development of gas-engines. Messrs. Beardmore have already published some particulars of their 500 and 1,000 h.p. vertical marine-engines, and I hope that some of their people will tell us something of the success of their engines when they have got them to work, and particularly as to the success or otherwise of the reversing-engines. That is one of the great problems—viz., how to adapt them successfully to marine work. They might be useful also to some of our members for driving rolling-mills. Then I added a few things in the paper as to how much is done in this direction in this country. The only other point is the question of cleaning the gas. I fear, as a builder of gas-engines, that a good many of our members connected with iron- and steel-works are troubled about the difficulty of getting gas-engines to work properly at first because of the imperfect cleaning of the gas. You will see by the illustrations given in the paper that there are many people who have been working and cleaning gas, and many of these users have obtained very satisfactory results. I noticed in the German paper it was stated that

gas is cleaned to the extent of 0.02 g. of impurity. I telegraphed to Mr. Cochrane yesterday, and the result was that he replied that ordinary blast-furnace gas was cleaned to 0.0025, which is as clean as the air in this room, or possibly cleaner, as we happen to be in London. The figures I received from Mr. Cochrane were 0.0025, 0.0026, 0.0033, 0.0037, 0.0025, 0.0024, 0.0024, 0.0019, etc. That justifies my statement that gas can be cleaned to 0.02, as described in the German and in my own paper. Therefore I hope you will understand that the great initial difficulties in cleaning the various waste-gases for use in gas-engines have effectually been overcome.

Mr. Greiner:—Mr. Theisen has built an apparatus of a special sort which is the type shown in Mr. Westgarth's paper. Our experience at Seraing some seven or eight years ago was that Mr. Theisen was wrong when he said that the air was warmest when it entered the apparatus, and that it gave the best results. That is not right. The air must come into the Theisen apparatus and generally into all centrifugal apparatus as cool as possible, and the reason is that when air is heated it holds more steam or vapor, and the more vapor there is in the air the more dust there is in it. For that reason it is better to cool the gas before it goes into the Theisen apparatus, and all centrifugal apparatus. We could clean 4 or 5 g. of dust quite well, but the result would be that after a few days or a few weeks the apparatus would become so very full that it would be difficult to clean it at all, because the dust is put on the ledge. Our experience is that the cooler the air the better it is for entering into the Theisen apparatus, and it gives the best results that way. That has been pointed out by Professor Osann, and it agrees absolutely with our experience at Seraing with the Theisen apparatus itself.

Julian Kennedy, Pittsburg, Pa.:—We in America are only beginners in the gas-engine business, so that we can have very little to say about it, although we hope to learn a great deal about it. We all appreciate that the great thing to be learned now, perhaps, is the easy and thorough method of cleaning gases, which has to be worked up. We have had very little experience of it in America. The Lackawanna Steel

Co. are about the only people who have done anything in gas-engines in America. They had some trouble at the start, as was only to be expected, but they are now achieving a great deal of success with their gas-engines, and the general feeling is that the era of this style of motor is dawning with us.

DR. R. W. RAYMOND, New York, N. Y.:—After more than 25 years of service as the Secretary of the Institute, I have learned that my knowledge of contemporary progress depends largely upon the members of my Society. If I do not get any papers on gas-engines, I may not know much about gas-engines; but when the members of the Society begin to talk them up, then I realize that a new era has dawned. In the Institute of Mining Engineers, which includes blast-furnace managers, this is beginning to take place. American ironmasters are coming to the gas-engine. In spite of the wealth of our rich ores, our half-developed mineral resources, and our youthful strength as a nation, we are beginning to learn that heat and the materials that yield heat must be economized. We are not so rich that we can go on wasting; and if we succeed at all in maintaining the position to which we have somewhat suddenly jumped in this branch of industry, it must be done by carrying our book-keeping out to the third place of decimals, as is done in the papers read to-day, and by saving as well as spending. I believe I may say for American metallurgists, and their great individual or corporate combinations of capital, that they have not only heaped up sums in investments which staggered the imagination, but have also made a single dollar go further by putting a great many individual dollars together;—which is indeed the only justification for our large accumulation of capital. Moreover, it is but fair, in an age when "trusts," and "combines," and great corporations get all the blame they deserve, and more, from other people, that those who have profited by them should speak an honest word in their defense. I would therefore say that in America, at least, as I presume also in England, the great concerns which install such engines as have been described here are the concerns which employ the best scientific aid, pay the best wages, do the best work, and most effectively serve the human race.

PROFESSOR WM. KENT, Syracuse University, Syracuse, N.Y.:—We have been greatly pleased in finding how the Germans are leading the way and the English also are beating the Americans in the adoption of gas-engines, but it seems to me to be a little maneuver of the Americans to watch how others would do the work and thus to get the value out of the experience of others without the expenditure of their own money. It was found in the days of street-car transportation in America that the man who was first in substituting electric transportation for the mule was the man who lost the most money, and that those who waited four or five years until those bold, enterprising persons had lost their money, and gained their experience—those who held on to the mules—ultimately got the money out of the other men's experience. So Americans have saved an immense amount of money by not building gas-engines. Now they are going to build gas-engines with licenses from the German and the British patentees, and they are going to do just as good work as the Germans. The Germans have developed the gas-engine and the cleaner, but there is a good deal yet to be done. The field for the gas-engine is, of course, to utilize the waste-gases of the blast-furnace; but the blast-furnace produces gases of irregular quality and quantity, and the demand of the blast-furnace for power is also irregular. What we need is a governing-apparatus with which to regulate the gas-engine plant, and that is suggested in Mr. Reinhardt's paper, in which auxiliary producers are mentioned, using coal to furnish the regulating supply, and to make up for the irregular demand. In the future there is going to be a large plant near a large city where iron-works are situated, and this will supply the electric light and the electric power to the city. Such a plant would have reserve gas-producers using bituminous coal direct, and burning up all the hydrocarbons in the producer itself. The producers mentioned in the paper (there are at least two of them patented in America, of one of which I happen to be the patentee) would do that. I believe that practice in the future will take the form of regulating the supply for the gas-engine plant by having a surplus available derived from the auxiliary gas-producers, and the installation will thus be complete for the utilization of blast-furnace gases and for the development of electric power for rolling-mills and for power purposes in the adjacent town.

E. J. DUFF, Liverpool, England:—All of us who have been interested in the development of gas-power and the new method of gas-production by the recovery of ammonia must have read Mr. Reinhardt's paper with great interest and advantage. It is a paper that goes very thoroughly into the matter and brings the subject right up to date. I will confine my remarks to one topic—viz., coke-oven gases. About seven years ago I was called upon to erect some by-product coke-ovens at Widnes. I found that when the ovens were put to work they produced a surplus of gas; that is to say, after the gas was put out of the ovens and burnt under the ovens there was still a surplus of gas which went to waste. There was also a good deal of surplus steam raised from waste gases—that is, from the burnt gases leaving the ovens; and after they had used what steam was required for the recovery-process there was still a surplus. I set to work to discover how to utilize the surplus gases after the coke-ovens had been fired. Eventually we decided to adopt gas-engines to utilize the gas to drive dynamos, and to run electric furnaces by the dynamos for the production of calcium carbide. That worked extremely well from the start. We calculated that we had enough gas to run 500 h.p. I applied to various makers for 500-h.p. engines, and none of them would build one. That was only seven years ago. I adopted 250-h.p. engines, the largest built in this country then. Very little scrubbing and washing was required for that gas, and the only alteration that I found necessary after commencing to run the engines was the insertion of a little box, 2 ft. square, of iron oxide, to take out a trace of sulphur and cyanogen. In that connection I find that many coke-ovens are working in this country producing sufficient cyanide to make it worth while to recover it. Then, again, with regard to the surplus steam coming from those ovens, Mr. Reinhardt said: "In the new regenerative coke-ovens the waste heat is utilized for pre-heating the oven itself, whereby there is an economy in gas, and a greater excess of gas is available for driving gas-motors." That, I think, is one of the best things for the recovery, inasmuch as it takes away that surplus steam, and it gives the excess of gas which we can utilize in gas-engines. Mr. Westgarth in his paper referred to the nature of the gas used in the engines tabulated in his paper, and he has thought a paper on

gas-engines would be incomplete without reference to gas-cleaning appliances. In this connection I would like to mention that the twenty-eight engines given in Table I. of the paper, with a total capacity of 32,600 i.h.p., are all working on some form of Duff producer-gas. I would also mention that Beardmore & Co. have built, and are building, producers of a total capacity of 100,000 h.p., all of which clean the gas and recover from it the by-products, the sulphate of ammonia, in sufficient quantity to pay the whole of the coal-bill. The greatest credit is due to Mr. Beardmore for his courage and ability in experimenting on such a large scale, and not only for undertaking the extensive use of gas and electric power in connection with the general work of shipbuilding and engineering, but also for applying the system to rolling-mills and steel-melting furnaces under the exacting and somewhat hard service required in steel-works. This has all been accomplished since I designed Mr. Beardmore's plant four years ago. None of these papers on gas-engines, so far as I am aware, have referred to a very important installation of gas-power which has been erected and put to work at Madrid, in Spain. The gas-plant is my own design, and the engines are of the Nürnberg pattern, of 2,000 h.p. There are six engines of 2,000 h.p., with a total capacity of 12,000 h.p. One of these engines has made a record run of practically six months, and there is no trouble with tar.

James Hamilton, Coatbridge, England:—The first feeling of manufacturers in this country on reading the papers is that of envy at the great development and the great progress which the Germans have made. It is not so much the fault of the British gas-engine makers as of the British gas-engineers. I think that the gas-engine in this country, now that it has got through its period of initiation, is going to develop more rapidly. It is a pity that the ironmasters have not considered the merits of the engine more, because there are great savings possible, and great savings have been effected where it has been tried. British gas-engines have not developed along quite the same lines as the German engines. The German engines have been designed chiefly with a view to increasing the maximum power, and not so much with a view to increasing the economy. The four-cycle engine described by Mr. Reinhardt is certainly

more economical than the two-cycle engine; but that is not altogether the reason for its being adopted in Germany and in Belgium. In this country a greater amount of work has been done with producer-gas than with blast-furnace gas, and in consequence the English engines have been developed rather with a view to economy, because when we get the blast-furnace gas for nothing a little extra consumption is not of so much consequence as when we have to pay for the fuel to produce it. In a blast-furnace gas-engine the saving as compared with a steam-engine is about five to one—that is, five times as much power from the gas-engine as from the steam-engine with the same amount of cost. In the producer-gas engine we do not get so much saving because there is the question of the efficiency of the producer to be deducted from the result. We have to combine the producer and the gas-engine, and the efficiency of the whole thing is the combined efficiency of the two. The Continental designs have been to a certain extent adopted in this country, and are now working in competition with British designs, and no doubt the future will decide which type is the best. Each has its merits, but so far the British designs have held their place in competition with the Continental designs that have been taken up and made in this country. As a gas-engine maker, I must thank the gentlemen who have so liberally placed the results of their experience and knowledge at the disposal of the Institute. They have conferred a great favor on the iron- and steel-industry generally, and the gas-engine making industry in particular, and I am sure we owe the authors our best thanks.

A. T. Tannett-Walker, Leeds, England:—As one interested in gas-engines, and who, thirty-four years ago, saw a good deal of the development of these motors, I would like to express my indebtedness to those who have read the papers. I consider the paper of Mr. Reinhardt quite a treatise on gas-engines, and we are all indebted to Professor Hubert and to our valued friend, Mr. Greiner, for the information they have given to us. We were told some eight years ago to what perfection we could work by blast-furnace gas-engines, and I have taken great pains to see whether they are perfect, but I find there is a great deal to do to make them perfect. As our American friend has said,

those who stuck to their mules have kept their money. But, of course, if there were no pioneers we would have no great inventions brought to perfection. There is one thing I must take exception to—viz., the remark of my American friend (Professor Kent), that all the developments have been made by the Germans. The Otto gas-engine was brought to this country and offered to my father, the late Benjamin Walker, who, however, I am sorry to say, did not take it up; but Crossley Brothers took it up and, to their credit be it said, they have developed it with certain inventions, such as the valve-gear and other innovations, the result of experience. Therefore I say that the Germans have largely developed gas-engines, but they were not the originators, they were not the pioneers of gas-engines. We must, in fact, give the old country, the fossil that Sir James Kitson referred to, the chief credit, because the old country has held its place even in the manufacture and the development of these modern motors.

Mark Robinson, London, England:—I have been engaged over the design and construction of large gas-engines lately, and it has proved to be a very interesting subject. It is a subject about which there is a great deal to learn, and I can cordially indorse the remarks of the gentleman from the other side of the Atlantic who has said there is a great deal of loss upon them. The German paper seems a most valuable one, and it is a mine of information. All the papers are very useful, and we owe great thanks to the authors for them. But there is one subject which these papers do not touch upon, and I wish they did. It is natural to keep the eye on the engines to use blast-furnace gases when addressing the Iron and Steel Institute; but allusion has been made to the use of the gas-engine for other purposes, and, of course, it is used for producer-gas. When the company with which I am connected took up the gas-engine it was almost entirely for the using of producer-gas and for the driving of dynamos; and from the best information we can gather we believe there are difficulties when it is working with producer-gas unless there is some very special arrangement for cooling the engine. I do not think that any allusion is made to the scavenge in any of those papers. I believe the great makers of the four-cycle engines, Messrs. Cockerill, do not

use the scavenge at all, and think it unnecessary. I believe it would be a very good thing for English designers to get some pronouncement from the great masters on the Continent as to whether the scavenging is necessary or not. On the other hand, they are almost always working with blast-furnace gas, which is very much cooler and is not liable to lead to pre-ignition and other troubles, as we might suppose producer-gas to be. It had been stated that some of the Continental gas-engines were worked with producer-gas, and if it were the case that they worked with gas of high calorific power and did not give any trouble, but operated successfully without a scavenge, it would be very useful to have details from persons in a position to speak with authority.

PROFESSOR TURNER, Birmingham, England:—I cannot speak as an authority on large gas-engines, as the gas-engines with which we have to do at our University at Birmingham are small. All that we have is an installation of the Mond gas-plant for engines, the largest being 150 h.p., used for the production of current for various purposes throughout the University. I need scarcely say that we have had no trouble with those engines, and they work with the greatest possible satisfaction. But the Institute is interested in the question of large gas-engines in connection with the manufacture of iron and steel, and many of us have been wanting information in connection with the development of these engines. The information is not that which we could get at the University, and it could only come from practical men. We are very much indebted to those who have been good enough to give it to us.

B. H. THWAITE, London, England:—I congratulate the authors of the three papers in providing a more or less complete record of the progress in the use of the blast-furnace gas-engine and its displacement of steam, and of the evolution of the high-power-capacity gas-engine. Apparently the energy that is already being developed is close on half a million horse-power. We should not allow an opportunity to pass without an expression of admiration for the splendid enterprise shown by engine-builders and ironmasters in Germany in risking the capital involved in raising the unit-power capacity of gas-engines.

The other day on formulating an electric-power generating-scheme I had no hesitation in specifying for 10,000 i.h.p. with multiple engines. I am glad that in the distribution of the palms the claims of our country have not been overlooked; but when the authors write again on the historic part, and especially in the light of the truth that is now being admitted without hindrance, I hope they will carefully read patent document No. 8,670, of May, 1894, also Professor Watkinson's paper read before the West of Scotland Institute on March 15, 1895, and the discussion thereon, with the remarks of Mr. James Riley, the then chairman.[1] If they do so they will then agree that the word "simultaneous" is absurd; and the reference to his propositions in *Le Rappel*, *Le Figaro*, and other French newspapers in 1885 was rather contradictory to the statement made by Mr. Hubert. To say that "the investigations were independent of Thwaite's experiments, which were not generally known on the Continent," was incorrect, for the experiments were practically the same as mine, which were widely known on the Continent in 1895. The English were first in the field. In my paper read before the British Iron Trade Association in 1898, I gave a thermal comparison between steam-power efficiencies of the ordinary iron-works type of the blast-furnace gas-engine as follows: Heat expenditure to secure 1 h.p. per hour of energy in steam iron-works plant equals B.t.u. ÷ 43,300. The blast-furnace gas-engine at that time developed the same power with a heat expenditure of one-fourth, or 10,328 B.t.u. To-day we can rely upon securing a still higher result of efficiency. Mr. Hamilton will guarantee to develop an indicated horse-power with an expenditure of 8,000 B.t.u., and so will Mr. Thomas Westgarth. I have explained in the *Times* that one reason why I have sought to develop power with gases of low calorific value is my recognition of the effect on gas-engine limits of efficiency of the law of increase of specific heat with increase of temperature. In the selection of such a gas I de-

[1] I am glad and it is only right to acknowledge that Mr. Andrew Carnegie was the first American to realize the value and importance of my work in harnessing the blast-furnace to the gas-engine. As his guest in Scotland in 1897, he acknowledged the economic value of my system; and but for the fact that the blowing-engines and boilers at the Edgar Thomson works, Pittsburg, had just been completely renewed, he (Mr. Carnegie) would have applied the system at least tentatively.

cided in favor of carbon monoxide, and the exclusion as far as practicable of hydrogen gas, for various reasons. A gas of the chemical constitution of blast-furnace gas gave me the power-gas I wanted, and you see the justification of my pre-1890 reasoning in the records of the three papers read to-day. It is pleasant and refreshing to find that Mr. Reinhardt's idea of a standard system of cleaning blast-furnace gas is practically that of the Thwaite system. Looking back over 13 years, it is certainly amusing to remember, although at the time the statements were made vexation and indignation were the dominant feelings, how we were once told that no cleaning of the furnace-gas was required, in distinct contradiction of my own experience and the formulæ of my 1894 patents. Mr. Reinhardt referred to a standard type of purifying-plant for blast-furnace gas, and observed: "The gases on leaving the blast-furnace are led through a series of so-called dry purifiers, and thence through long pipe-lines into the coolers or scrubbers, and from these into the so-called centrifugal purifiers (Theisen apparatus or fans with water-spray). After leaving the above plant the purification of the gas should be complete, so that before being admitted into the engine the gas has only to be dried in filters or in capacious tanks." It has taken all these years to admit the sequence of my 1894 patent to be correct. At one time (1899) we were told that no cleaning was necessary. I know that Mr. Greiner has withdrawn that statement, but for a certain number of years it was accepted. In the Thwaite system, non-electric, we have under suction-influences, first, the rough hydraulic cleaning and cooling of the gas; secondly, the air-cooling of the gas; thirdly, the application of centrifugal influences under combined suction- and pressure-effects; fourthly, coarse filtration under pressure; fifthly, fine filtration under pressure; sixthly, establishment of a constant pressure. Here I might remark that the centrifugal cleaning efficiency is in proportion to the temperature—the cooler the gas as it enters the centrifugal instrument the more effective the cleaning efficiency. Any variation from my original cleaning-plant is, I consider, at the cost of efficiency and economy, and has been prompted more by a desire to evade patent-rights than to secure improvement. I am glad Mr. Reinhardt supported the use of a pressure-governor holder, which has been an essential feature

of my system from its practical inception. Reading between the lines of Mr. Reinhardt's peroration we are almost justified in assuming that the maintenance of German iron-trade prosperity is largely due to the application of blast-furnace gas-engines in German works. But is it not a fact that the splendid enterprise is a product of the continuous German prosperity, proved by the unbroken line of the curve of increased pig-iron output, which has risen far higher than that representing the progress of our iron trade? Although the engines described show well-appreciated progress, and the attained stage of perfection justifies the concentration of enormous powers in one unit of grouped cylinders, I hope that the workers in the field of gas-engine design will persevere in attempts to secure still further triumphs.

Heat-Treatment of Steels Containing Fifty Hundredths and Eighty Hundredths Per Cent. of Carbon.

Discussion of the Paper of C. E. Corson, p. 388.

ALBERT SAUVEUR, Cambridge, Mass. (communication to the Secretary*):—On close examination I think it will be found that the evidence by which Mr. Corson claims to have shown the inaccuracy of a statement I made a few years ago, is far from convincing. This statement is:

> "Hot work as such has no influence upon the structure of the metal. Indirectly, however, by retarding crystallization until a lower temperature is reached, it may influence its structure most decidedly; but the same results could be accomplished by heat treatment alone, *i.e.*, by re-heating the unworked metal to the temperature from which the worked piece was allowed to cool undisturbedly." *The Metallographist*, vol. ii., p. 267 (under the head of "Changes of Structure Brought About by Work").

Of the two pieces of steel tested by Mr. Corson, one was worked and finished at a "cherry-red" and the other reheated to a cherry-red, and the assumption was made that in both cases the temperature was about 715° C., although no pyrometric device was used to record it. The possibility of considerable difference in temperature alone is so great as to invalidate Mr. Corson's inferences. The apparent temperature,

* Received September 12, 1906.

moreover, is so close to the critical point of the steel as to render any conclusion very hazardous. It is not at all evident that the annealed piece was re-heated past its critical point, in which case re-heating would have had practically no effect. Nor is it evident that the piece forged and finished at a cherry-red was not actually below the critical point, which, by definition, I have called cold worked.

Mr. Corson's explanation that "Under proper re-heating, on the other hand, the steel becomes a solid solution, from which crystals of approximate homogeneity and uniform size may separate," will not be readily understood.

It should be borne in mind that my statement refers to hot work only; that is, the work performed above the critical point, and in that range the steel is in the condition of a solid solution just as well as if it had been re-heated to that temperature. In other words, if a piece of steel be worked to 800° C., and allowed to cool slowly, or if it be re-heated to 800° and slowly cooled, it will in both cases cool undisturbedly from the condition of a solid solution.

In view of the above explanation, the statement which Mr. Corson criticises cannot be set aside in the light of a single experiment, especially since that experiment is so crude, and the results so uncertain, as to cause its significance to dwindle into insignificance.

Aside from the discussion, Mr. Corson should be congratulated on the excellence of his photomicrographs.

INDEX.

[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. References to papers expressly treating of the subject named are likewise in italics; and casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

ERRATA.

Page.	Line.	
31	3	For "quantity of tons" read "quantity of slime."
35	10	Expunge "recorded in Table VII."
37	24	Expunge "to."
38	33	For "page 89" read "page 9."
44	40	For "page 86" read "page 6."
47		In table, third line under heading "*Excavators and Conveyors,*" expunge "per 40 ft. of."
52		Second line under table. For "32" read "0.32."
90	16	For "Sargent" read "Sergeant."
147		Fig. 2, title. For "White Horse Claim" read "Wild Horse Claim."
154	19	For "porlarizing" read "polarizing."
173	35	For "character" read "characteristics."
178		Under title, add "(Bethlehem Meeting, February, 1906.)"
212		Table VII., second line. For "1.5" read "1.0."
389		Footnote [1], fifth line. For "unworked" read "worked."
441		In place of paragraphs (9) and (10) read: "(9) Two Harz jigs on 1.25-in. feed. Concentrates to (12), middlings to (13) through (11), (7), (8), hutch-product to (14). "(10) Four Harz jigs on $\frac{7}{8}$-in. feed. Concentrates to (12), middlings to (16) through (15), (7), (8), (13), hutch-product to (14)." Paragraphs (14), (17) and (19). For "(35)" read "(20)."
442		Paragraphs (27), (30) and (33). After "(32)" add "through (31)."
526	16	For "*mustin*" read "*Austin.*"

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